MINING METHODS AND COSTS--
LA SAL MINING & DEVELOPMENT CO.
LA SAL URANIUM MINE, SAN JUAN COUNTY, UTAH

By W. L. Dare and D. T. Delicate

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by

W. L. Dare[1/] and D. T. Delicate[2/]

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SUMMARY AND INTRODUCTION

This publication is one of a series by the Federal Bureau of Mines describing methods and costs of mining uranium on the Colorado Plateau.

The La Sal mine is in the Big Indian mining district, about 29 miles southeast of Moab, San Juan County, Utah. The claims that encompass the mine were located in March 1953, and the La Sal ore body was discovered as a result of a drilling program begun in August 1953. The mine is owned by the La Sal Mining & Development Co., a subsidiary of the Homestake Mining Co. Homestake continued the drilling program and in May 1954 began sinking a 572-foot shaft and constructing a surface plant. The plant and shaft were completed in October 1954.

The La Sal ore body is in the Chinle (Triassic) formation. It is tabular and dips with the enclosing beds about 8° southwest. The ore body is generally 4 to 8 feet thick but in a few places is as much as 20 feet thick. It is being developed by a series of strike drifts and dip inclines, driven within the ore stratum on about 100-foot centers. A haulage level driven beneath the ore body is connected by raises to the upper workings.

The waste rock that comprises the back of the ore body lacks the strength and rigidity to permit a standard room-and-pillar method of mining to be used. Therefore, experimental mining was begun in March 1956, and a method was developed that permits almost complete extraction of each block of ore, after which the back is induced to cave. It is a retreating room-and-pillar method, based on the removal of 18-foot panels, and has proved successful under conditions that are encountered in the La Sal mine.

ACKNOWLEDGMENTS

The authors wish to thank Dr. Donald H. McLaughlin, president, Homestake Mining Co., for permission to write this paper. The cooperation and assistance given by the staff of the La Sal Mining & Development Co. are gratefully acknowledged: Gordon M. Miner, assistant superintendent of mines; Paul L. Jones, engineer; Donald Weierman, mine foreman; Henry B. Deton, accountant; Donald Nelson, personnel; and Dean T. Crossland, sampler, who were helpful in providing details and data on the mining operation.

LOCATION AND PHYSICAL FEATURES

The La Sal mine is in sec. 33, T. 29 S., R. 24 E., and secs. 27, 28, 32, 33, and 34, T. 29-1/2 S., R. 24 E., Salt Lake meridian, San Juan County, Utah (fig. 1). The mine is reached from Moab, Utah, by traveling southeast on United States Highway No. 160 for 22.1 miles; east on newly paved Utah State Highway No. 46 for 6.6 miles; south on Big Indian Wash improved access road for 4.9 miles; right, following the road south to the rim bordering the west side of Big Indian Wash, for 0.8

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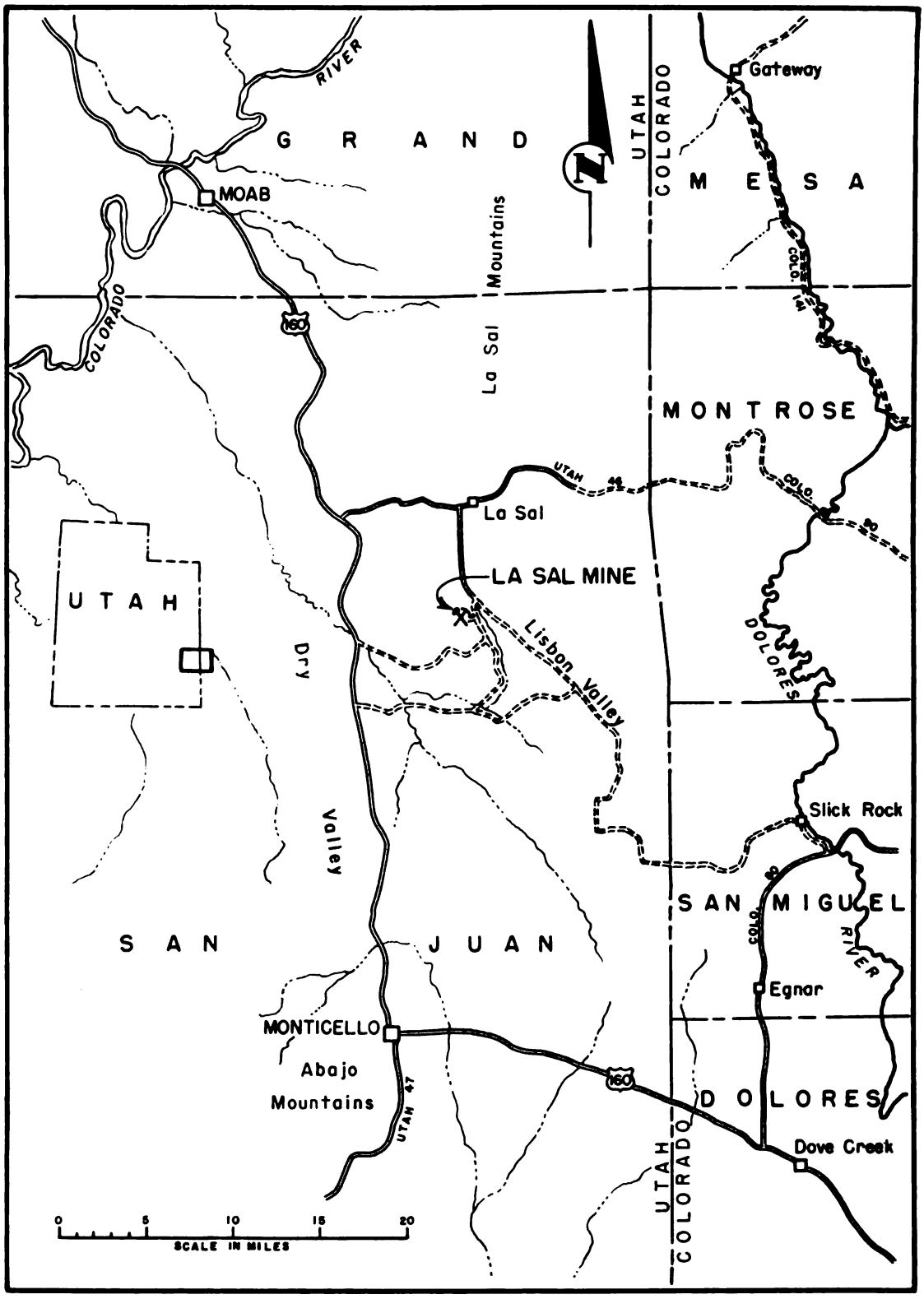

Figure 1. - Location map, La Sal uranium mine, La Sal Mining & Development Co., San Juan County, Utah.

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mile; and, finally, west an additional 0.5 mile to the mine. The shaft is in one of the upper tributary canyons of Sandstone Draw, which carries drainage westward. The altitude of the shaft collar is 6,809 feet (fig. 2).

The topography of the area is rough. Erosion has cut the sedimentary formations into a disorderly and complex drainage pattern. Steep-walled canyons and washes dissect the higher areas, leaving mesas walled by cliff-forming sandstones. Within 1 mile of the La Sal shaft, altitudes range from 6,400 to 7,200 feet. The south end of the La Sal Mountains is about 10 miles north of the mine, and Mount Peale rises to 12,721 feet above sea level.

The climate is arid. Summer rains are mostly heavy downpours, which sometimes cause local flash floods. Winter snows usually average 6 inches but may accumulate to as much as 2 feet. Temperatures rise to above 100° F. in the summer and fall to below freezing during the winter. Water for mining and camp use is hauled from local wells.

Vegetation is sparse and predominantly juniper, pinon pine, and sage. No timber that can be used for mining purposes grows in the Big Indian Wash district.

HISTORY AND PRODUCTION

The earliest known mining in the area began around 1904 in the old Dry Valley mining district which is about 9 miles southeast of the La Sal mine. The history of mining in the Dry Valley district follows the general history of uranium mining on the Colorado Plateau. Initially, the carnotite ores in the Salt Wash sandstone member of the Morrison (Jurassic) formation were mined for their radium content. Mining for vanadium was intermittent from the First through the Second World Wars. Mining during the current period was begun about 1946, principally for uranium.

The first production of ore from the Big Indian Wash area was recorded in 1908, when oxidized-copper ores, occurring in sandstones and conglomerates of the undifferentiated Dakota-Burro Canyon (Cretaceous) formations, were mined from a number of small deposits along the downthrown side of the Lisbon Valley fault. The more important producers were the Big Indian mine, about 1 mile northeast of the La Sal mine, and the Black Bird mine, about 10 miles southeast. Copper ore was produced last from the Big Indian mine by the Ohio Copper Co. during the Second World War.

The earliest known production of uranium-vanadium ores from the Chinle (Triassic) formation in this area was in 1917 and came from the Serviceberry and Divide mines, which are on the upthrown side of the Lisbon Valley fault, about 10 miles southeast of the La Sal mine. Sporadic production from 1948 to 1955 came from a number of small mines in the Cutler (Permian) formation at its outcrop along the escarpment.

It was not until July 1952 that prospecting in the Big Indian Wash district began in earnest. The first major discovery of uranium ores in the Chinle formation was made on the Mi Vida claim by Charles A. Steen, when his diamond drill cut through 14 feet of high-grade uranium ore. Soon afterward, exploratory drilling was begun by many firms and individuals willing to risk money on wildcat drilling in search of totally hidden ore bodies. Uranium ore was first produced from Steen's Mi Vida mine December 6, 1952.

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Figure 2. - Surface plant, La Sal mine.

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The claims now owned by the La Sal Mining & Development Co. were located in March 1953 by Frank Richardson of Ouray, Colo., T. C. Hudson of Pawhuska, Okla., and their associates. They organized the La Sal Mining & Development Co., which began an exploratory-drilling program in August 1953. The La Sal ore body was discovered as a result of this program. The Homestake Mining Co. was granted an option March 17, 1954, by the La Sal Co. to purchase all of its outstanding shares. This option was exercised in May 1954, giving Homestake control of the La Sal mine.

The La Sal Mining & Development Co. continued drilling through 1954, although a major part of the program had been completed by May 20, 1954.

The Centennial Development Co., of Eureka, Utah, was awarded a contract to sink the shaft and construct the mine surface plant. Bulldozing was started at the plant site May 20, 1954. Shaft sinking was begun July 27, 1954, and the shaft was bottomed October 27, 1954. The first ore produced from development was shipped in January 1955.

LABOR AND LIVING CONDITIONS

One of the major problems confronting mining companies in the Big Indian mining district is labor turnover, due mainly to inadequate housing, lack of nearby schools, and remoteness of the job. The La Sal Mining & Development Co. established a camp about 1/2 mile southwest of the mine, which provides living quarters for about 32 employees. Most of the men live in the company-owned bunkhouse or with their families in company-owned or privately owned trailers. A few miners live in Moab and commute daily. The company operates a boardinghouse and charges $95 a month for board and room. The management rents trailers to employees with families for $45 a month, which includes electricity, water, fuel oil, and use of the laundry and other camp facilities. Those living in privately owned trailers are charged $17.50 a month for the same conveniences. Water for camp use is hauled from a local well and piped to each housing unit. Grade-school children are carried by bus to and from the county school at La Sal, 8.5 miles from camp. The company has built a few permanent-type homes at La Sal for staff personnel.

Because of the high wages paid in the district, the labor supply is ample, but the turnover is high.

GENERAL GEOLOGY AND DESCRIPTION OF DEPOSIT[3-6/]

General Geology

The Lisbon Valley anticline and fault are the prominent structural features of the area. The Big Indian mining district lies along a belt that generally parallels these features. The anticline is a salt structure, which strikes northwest. Lisbon

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3/ Dix, G. P., Jr., The Uranium Deposits of Big Indian Wash, San Juan County, Utah: AEC RME-4022, 1953, 15 pp.

4/ Steen, C. A., Dix, G. P., Jr., Hazen, S. W., Jr., and McLellan, R. R., Uranium-Mining Operations of the Utex Exploration Co. in the Big Indian District, San Juan County, Utah: Bureau of Mines Inf. Circ. 7669, 1953, 13 pp.

5/ Lekas, M. A., and Dahl, H. M., The Geology and Uranium Deposits of the Lisbon Valley Anticline, San Juan County, Utah; Geology and Economic Deposits of East Central Utah: Intermountain Assoc. Petrol. Geol. Ann. Field Conf. No. 7, 1956, p. 161.

6/ Oral communication with AEC geologist, AEC camp, Big Indian Wash, San Juan County, Utah.

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Valley occupies the collapsed portion of the anticline, and Big Indian Wash is on the southwest flank. The fault strikes northwest, parallel to the axis of the anticline, and passes through the north end of the wash. The fault displaced the Hermosa limestone (Pennsylvanian) on the southwest side, so that it contacts the Dakota sandstone (Cretaceous) on the northeast side of the fault. The fault zone is mineralized with copper.

Formations in the Big Indian mining district range from Hermosa limestone (Pennsylvanian) to Mancos shale (Cretaceous) (fig. 3). Formations exposed along the escarpment that forms the west side of Big Indian Wash are the Cutler (Permian) and Chinle and Wingate (Triassic). Immediately east of the escarpment the beds dip as much as 22° southwest; the dip decreases to the west; and in the vicinity of the La Sal mine the average dip of the formations is 8°.

The Cutler formation in the area is about 880 feet thick. It is composed of fine-grained sandstone, with discontinuous lenses of coarse-grained arkosic sandstone, siltstone, and mudstone and is essentially red or red-brown. Small uranium-ore bodies, occurring in the arkosic sandstone lenses, have been mined at the old Big Buck, Small Fry, and Purple Paint mines, which are along the escarpment. The Chinle formation is about 400 feet thick and can be recognized by the slopes it forms. It is variable in color and consists predominantly of red and green mudstones and siltstones, interbedded with conglomerates and fine-grained, crossbedded sandstones. The Wingate formation is a massive orange-red cliff-forming sandstone, 350 feet in thickness, which caps most of the higher ground west of Big Indian Wash.

The Chinle ore bodies in the vicinity of the La Sal mine occur along a definite trend, which strikes about N. 23° W. Generally these ore bodies are elongated, and their longer dimension parallels the trend.

Description of Deposit

The La Sal ore body occurs at the base of the Chinle formation, within the ill-defined basal member called the Moss Back sandstone. It is tabular, has an average strike of about N. 21° W., and dips about 8° southwest.

The Moss Back member is predominantly gray and greenish-gray sandstone, with lenses of green mudstone, siltstone, and conglomerates. Ore occurs within the bottom 20 feet of the Moss Back member and, for the most part, is within 2 feet of the erosional contact with the Cutler formation. The ore often extends to the contact; in a few places slight mineralization has been found to extend a maximum of 6 inches into the Cutler. The ore stratum is predominantly sandstone, 3 to 10 feet in thickness often coarse-grained and cemented with calcium carbonate. This sandstone is interbedded with lenses of mudstone, siltstone, and conglomerate. The mudstone and siltstone lenses are thickest near the top of the sandstone. Mudstone pebbles are distributed through most of the sandstone, with minor amounts of carbon trash distributed principally within the mudstone seams.

The grade of the ore is fairly uniform throughout the major portion of the ore body, except for a few places with irregular masses of waste. Uranium mineralization seems to occur in both well-sorted and poorly sorted sandstone. Mudstone seldom contains uranium mineralization when at the top or base of the sandstone but often is mineralized when it occurs as a thin seam within the ore-bearing sandstone. The host sandstone generally is better cemented and harder where the U_3O_8 content is higher. At the western and northwestern limits of the ore body the ore sandstone thins considerably until the stratum is predominantly mudstone. Here a rather sharp cutoff exists between ore and waste.

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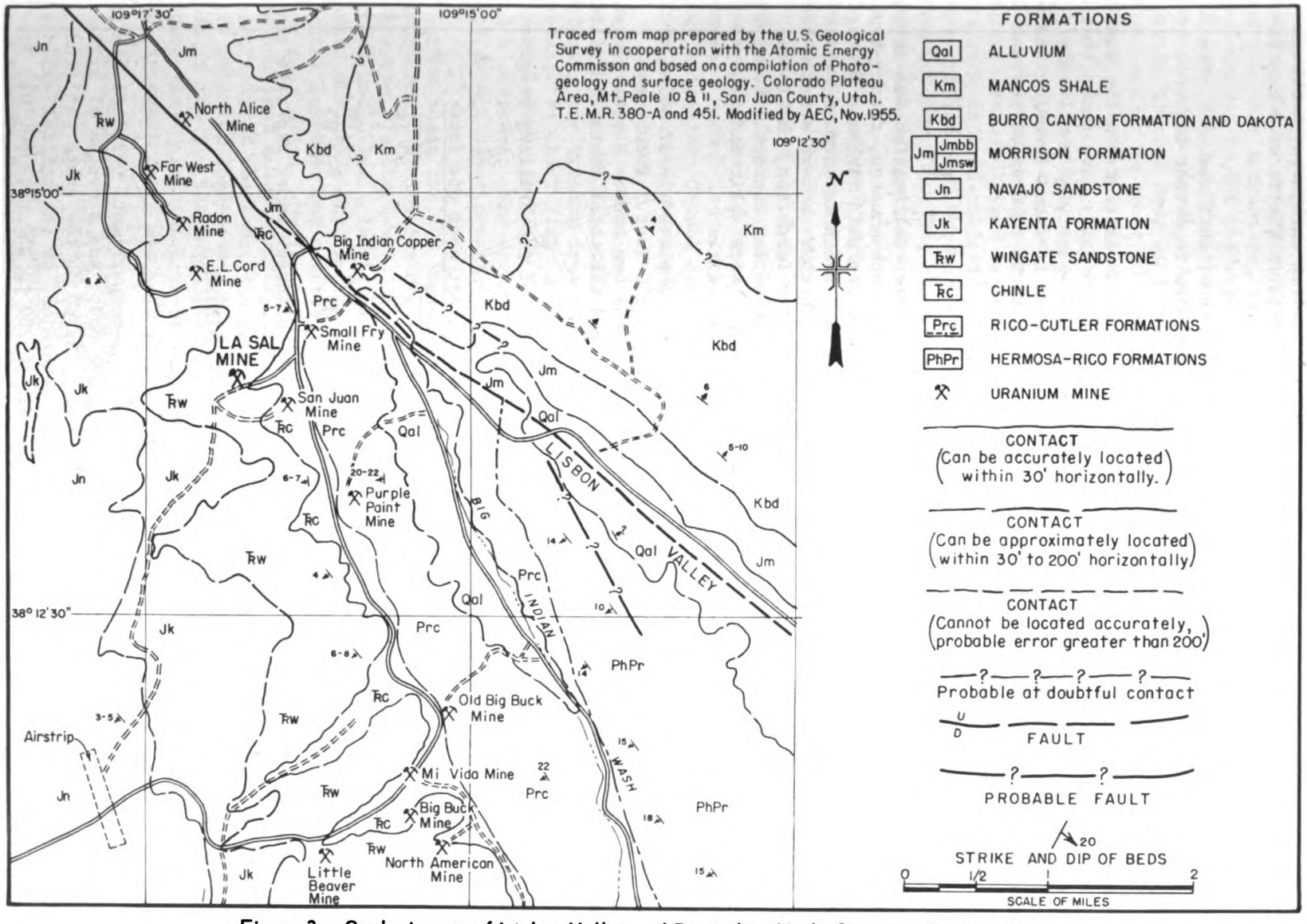
Traced from map prepared by the U.S. Geological Survey in cooperation with the Atomic Energy Commisson and based on a compilation of Photo-geology and surface geology. Colorado Plateau Area, Mt. Peale 10 & 11, San Juan County, Utah. T.E.M.R. 380-A and 451. Modified by AEC, Nov. 1955.
FORMATIONS
Qal ALLUVIUM
Km MANCOS SHALE
Kbd BURRO CANYON FORMATION AND DAKOTA
Jm Jmbb Jmsw MORRISON FORMATION
Jn NAVAJO SANDSTONE
Jk KAYENTA FORMATION
Trw WINGATE SANDSTONE
Trc CHINLE
Prc RICO-CUTLER FORMATIONS
PhPr HERMOSA-RICO FORMATIONS
URANIUM MINE
CONTACT (Can be accurately located within 30' horizontally.)
CONTACT (Can be approximately located within 30' to 200' horizontally)
CONTACT (Cannot be located accurately, probable error greater than 200')
Probable at doubtful contact
U D FAULT
PROBABLE FAULT
20 STRIKE AND DIP OF BEDS
0 1/2 1 2 SCALE OF MILES
109°17'30"
109°15'00"
109°12'30"
38°15'00"
38°12'30"
North Alice Mine
Far West Mine
Radon Mine
E.L. Cord Mine
Big Indian Copper Mine
Small Fry Mine
LA SAL MINE
San Juan Mine
Purple Paint Mine
Old Big Buck Mine
Mi Vida Mine
Big Buck Mine
North American Mine
Little Beaver Mine
Airstrip
LISBON VALLEY
BIG INDIAN WASH

Figure 3. - Geologic map of Lisbon Valley and Big Indian Wash, San Juan County, Utah.

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A large section of the La Sal ore body is 4 to 8 feet in thickness, although in a few places the ore thickens to 20 feet. The ore mined in the first 2 experimental stopes averaged 8.1 feet in thickness.

The rock formation overlying the ore is weak; therefore, this weakness, the thickness of the ore, and its undulating top and bottom were important features considered when the mining method was being chosen.

The ore mineral is uraninite, occurring as disseminated particles in the sandstone. Petrographic studies by the Atomic Energy Commission (AEC) have shown that the uraninite occurs interstitially with the quartz grains, partly or completely replacing the original calcite-clay cement. Uraninite also is found invading and partly replacing feldspars along cleavage partings. The ratio of uranium to vanadium is about 14:1.

METHOD OF EXPLORATION

The search for the totally hidden ore body and its later delineation was accomplished by drilling vertical holes from the surface with truck-mounted, rotary-type drill rigs. Exploratory drilling was done under contract and paid for on a footage basis. Tricone-type drill bits were used to drill to the favorable stratum, which then was cored with NX-size diamond bits. To make certain that the ore stratum had been penetrated and to check for possible ore in the Cutler formation, the holes were drilled approximately 10 feet deeper than the Chinle-Cutler contact. Cores were logged and radiometrically scanned; mineralized cores were split and assayed chemically. Holes were probed with a deep-hole recording unit.

To define the ore body, 19,615 feet of drilling was completed in 37 holes. Drilling depths ranged from 440 to 690 feet and averaged 530 feet. Twenty-two additional holes were drilled in ground outside of the ore body: Depths ranged from 479 to 808 feet; the average depth was 662 feet; and the total footage drilled, 14,555 feet.

Contract-drilling costs per foot for the latter part of the drilling program were as follows:

Depth interval, feet	Contract cost per foot	
	Noncore	Core
0-250	$2.10	$3.10
250-500	2.35	3.35
500-750	2.60	3.60
Over 750	2.85	3.85

Roads were built and drill sites prepared with a bulldozer under separate contract at $10 an hour.

METHOD OF DEVELOPMENT AND MINING

Development

The La Sal ore body is developed by a 572-foot, 3-compartment, vertical shaft; a haulage level driven below the tabular, dipping ore body; and a system of strike drifts and dip inclines driven within the ore body. Based on the shape, attitude, thickness, grade and tonnage of the ore body, as determined by the initial exploratory-drilling program, the production rate was set at 200 to 250 tons per day from a 2-shift operation.

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The site chosen for the shaft was the one considered best both as to topography and for future underground development. The shaft was sunk from a small canyon that was the most accessible, required the least excavation for a surface-plant site, and was nearest to the underground objective. It intersects the projected ore stratum 120 feet downdip from the lowest end of the ore body; thus the grades of all haulageways favor the loads. Although underground water has caused no serious problem, tapping the ore body at its lowest part permits drainage to the shaft sump. The shaft collar is about 20 feet above the canyon bottom, which insures against damage by summer flash floods.

Shaft sinking started in June 1954. A clamshell bucket on a truck-mounted crane was used to remove rock from the first 35 feet of shaft. After July 27, 1954, when the headframe and hoist installations were completed, a Riddell shaft mucker was used; this was equipped with a 3/8-cubic-yard clamshell, which loaded the rock into 35-cubic-foot-capacity sinking buckets. Twenty-hole bench rounds were drilled with Gardner-Denver S-58 rock drills. The shaft was bottomed at 572 feet October 27, 1954; the job was completed in 77 three-shift working days. Unit shaft-sinking costs are summarized at the end of this report.

The shaft has an 8-1/2- by 18-1/2-foot rock section. It is timbered with shaft sets and has 3 equal-size compartments, 5 by 6 feet in the clear; 2 are used for hoisting and 1 as a manway. The shaft sets, of conventional design, are built of 8- by 8- and 6- by 8-inch untreated Douglas fir. The skip guides are 4 by 6 inches, and the shaft is lagged with 2-inch red pine. Figure 4 shows isometric views of shaft timbering and framing details. Four sets of 3/4-inch hanging rods were used per set; manway ladders are staged every 2 sets. A 6-inch compressed-air line, 2-inch waterline, 2-inch sump-pump line, 20-inch ventilation pipe, 440-volt armored power cable, and a telephone line are hung in the manway compartment.

The shaft serves only the main haulage level, which is 519 feet below the collar. The shaft station was excavated in the form of a T, with the haulage drift entering the station from the southeast and extending 50 feet beyond it to form the crossbar of the T (fig. 5). For a distance of 50 feet on each side of the shaft, the haulageway is 10 by 7 feet in cross section. A trench, about 4-1/2 feet wide and 6 feet deep, was excavated along the shaft side of the station track. It is used as a slusher pocket.

The shaft station was excavated in the Chinle formation, principally in mudstone that overlies the ore stratum. The Chinle-Cutler contact is about 15 feet below the station level. To prevent the mudstone from sloughing, both timber and gunite were used. In the central part of the station, where the back slopes upward toward the shaft, the mudstone has been held successfully by 3-inch lagging placed tightly together and secured to the back with 8- by 8-inch timbers, pinned with 8-foot roof bolts. Gunite sprayed on the rock approximately one-quarter inch thick has provided a satisfactory seal; but, if the mudstone is soft and shattered, expecially at the back, the gunite will crack and fall. Conventional-type timber sets with top lagging are used in the shaft station in the part that forms the stem of the T. The horizontal area of the shaft station is approximately 1,600 square feet. A locomotive-battery-charger unit, drill-steel racks, a bit grinder, and equipment parts are kept in the station.

Development within the ore body was influenced by a number of factors. Only limited exploratory drilling had been completed before the shaft was sunk, and, rather than attempt to further define the ore body by additional drilling from the surface, it was decided that a systematic program of underground development would explore the ore body, establish its limits, and at the same time prepare it for a method of retreat mining. It was thought that development would:

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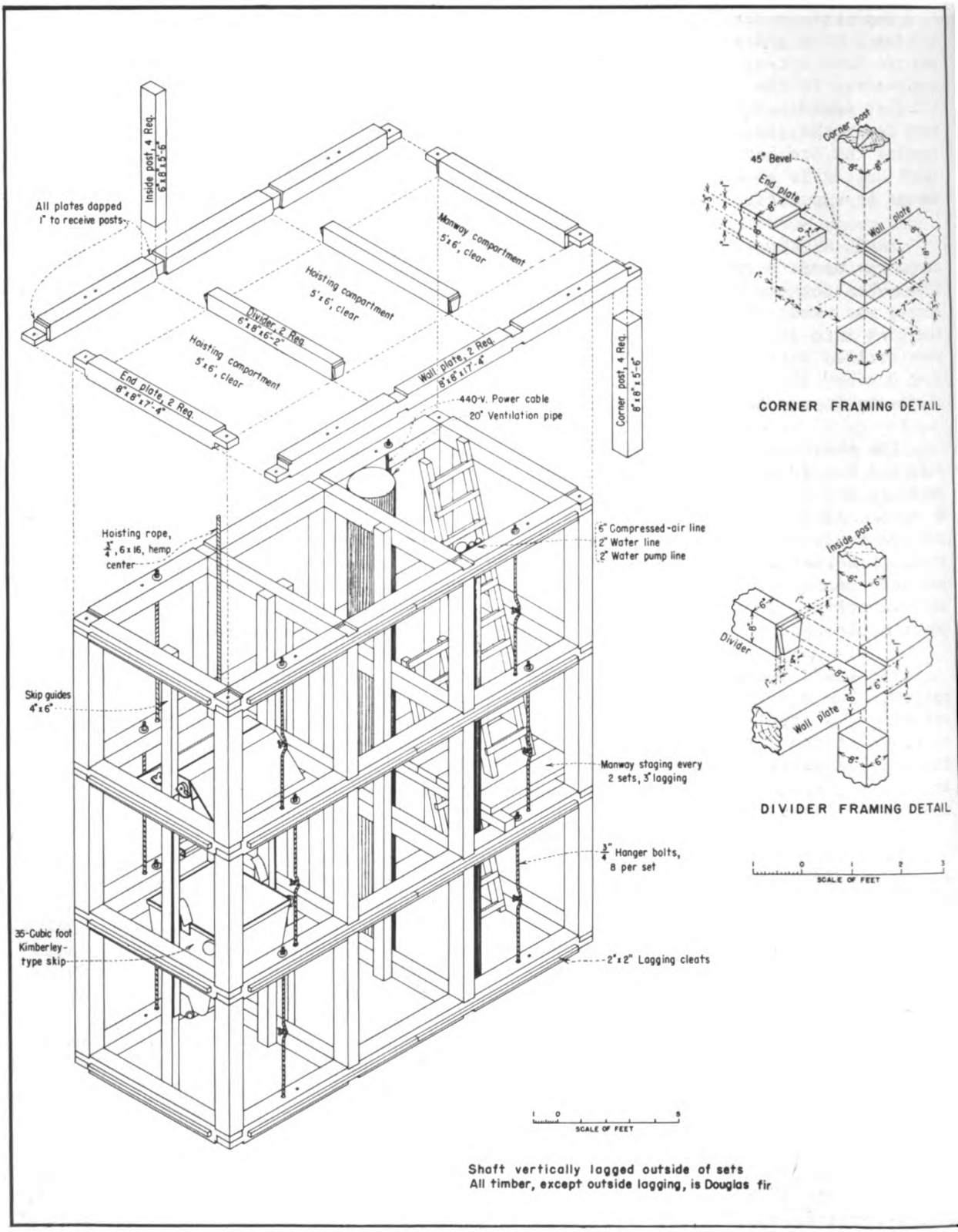

Figure 4. - Timber sets in three-compartment shaft.

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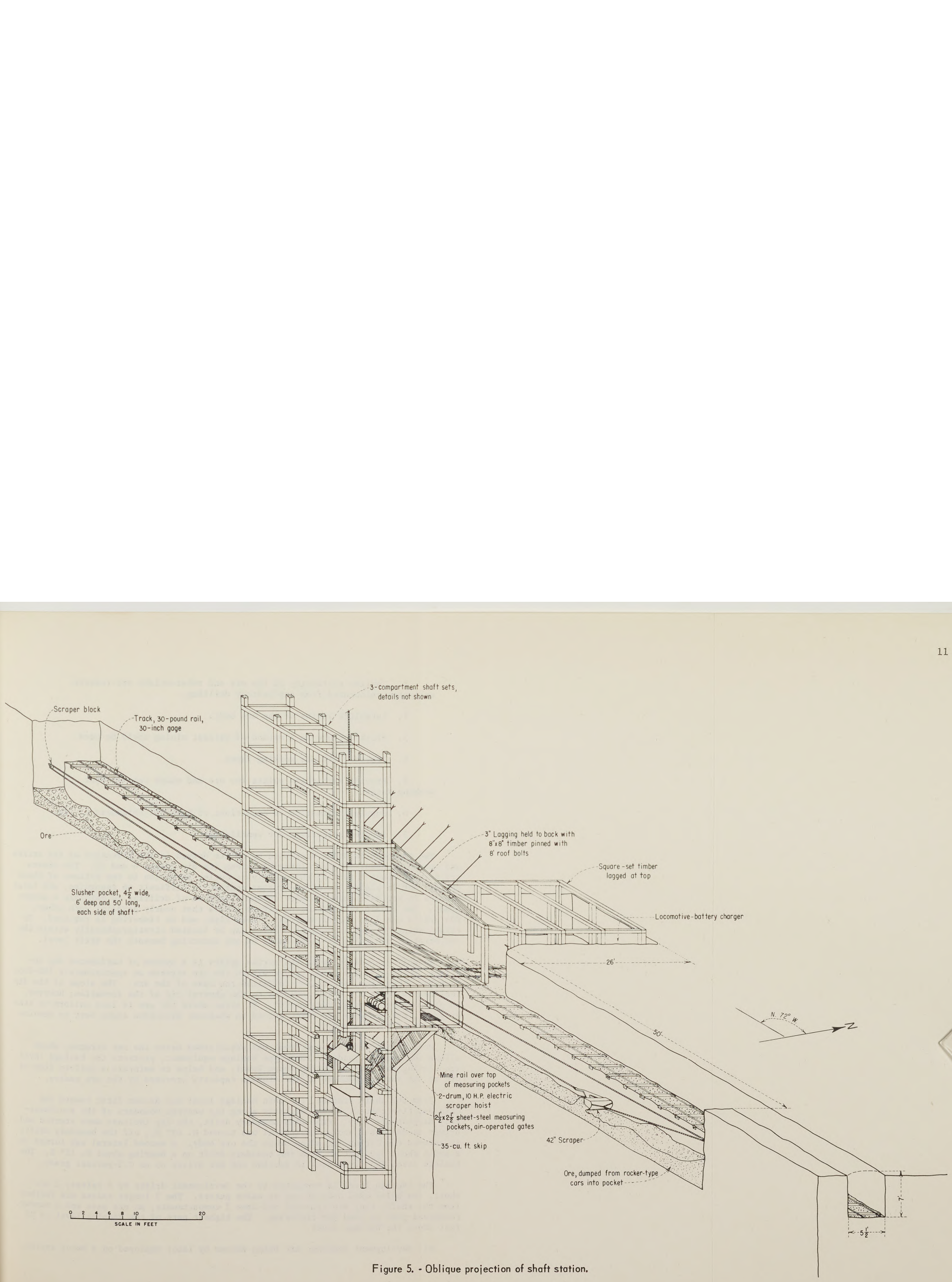
3-compartment shaft sets, details not shown
Scraper block
Track, 30-pound rail, 30-inch gage
Ore
3" Lagging held to back with 8"x8" timber pinned with 8' roof bolts
Square-set timber lagged at top
Slusher pocket, 4½' wide, 6' deep and 50' long, each side of shaft
Locomotive-battery charger
26'
N. 72° W
50'
Mine rail over top of measuring pockets
2-drum, 10 H.P. electric scraper hoist
2½'x2½' sheet-steel measuring pockets, air-operated gates
35-cu. ft. skip
42" Scraper
Ore, dumped from rocker-type cars into pocket
7'
5½'
0 2 4 6 8 10 20
SCALE IN FEET

Figure 5. - Oblique projection of shaft station.

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1. Confirm continuity of the ore and substantiate ore-reserve estimates calculated from exploratory drilling.

2. Establish limits of the ore body.

3. Block out ore so a method of retreat mining could be used.

4. Provide multiple working faces.

5. Provide extraction points for ore and waste removal near working faces.

6. Supply information on conditions of back and thickness of ore.

7. Provide aircourses for ventilation.

The ore body is being developed by drifts driven in the direction of its strike and spaced at about 100-foot intervals along its dip (figs. 6 and 7). The miners try to keep about 1 foot of the underlying Cutler formation in the bottoms of these drifts; but in places the Chinle-Cutler contact is difficult to recognize, and local changes in strike require changes in the direction of drifting to maintain a proper grade. The result is a series of strike drifts that nearly parallel each other. The drifts are 7 to 8 feet wide and 7 feet high, and no timbering is required. By establishing the Cutler contact, drifts can be located stratigraphically within the ore stratum, and there is less chance of ore occurring beneath the drift level.

Roughly perpendicular to the strike drifts is a system of untimbered dip inclines, which also are being driven within the ore stratum on approximately 100-foot centers. They, too, are driven to follow the base of the ore. The slope of the dip inclines is about 8°, which conforms to the general dip of the formation; however, in places they slope as much as 16°. In areas where the ore is less uniform in size or grade, the headings are being turned in whatever direction seems best to develop that part of the ore body.

Most of the main haulage level was established below the ore stratum, which allows the use of standard track-type haulage equipment, protects the haulage level from caving that occurs on the stope level, and helps to maintain a uniform flow of ore to the surface because of the storage capacity provided by the ore passes.

From the shaft station, the main haulage level was driven first toward the Chinle-Cutler contact, then northerly along the western boundary of the southwesterly dipping ore body. From this boundary drift, the dip inclines were started and driven as described above. A lateral was turned N. 63° E., off the boundary drift; it entered the Cutler formation below the ore body. A second lateral was turned at a point about 300 feet in from the boundary drift on a bearing about N. 15° E. The haulage level is 7 by 8 feet in section and was driven on an 0.5-percent grade.

The haulage level is connected to the development drifts by 5 raises; 2 are short, raw holes used only as ore or waste passes. The 3 longer raises are farther from the shaft; they are timbered and have 2 compartments, an ore pass and a manway. Ponderosa pine is used for timbering. The highest part of the stoping level is 85 feet above the haulage level.

All development headings are being driven by labor employed on a bonus system.

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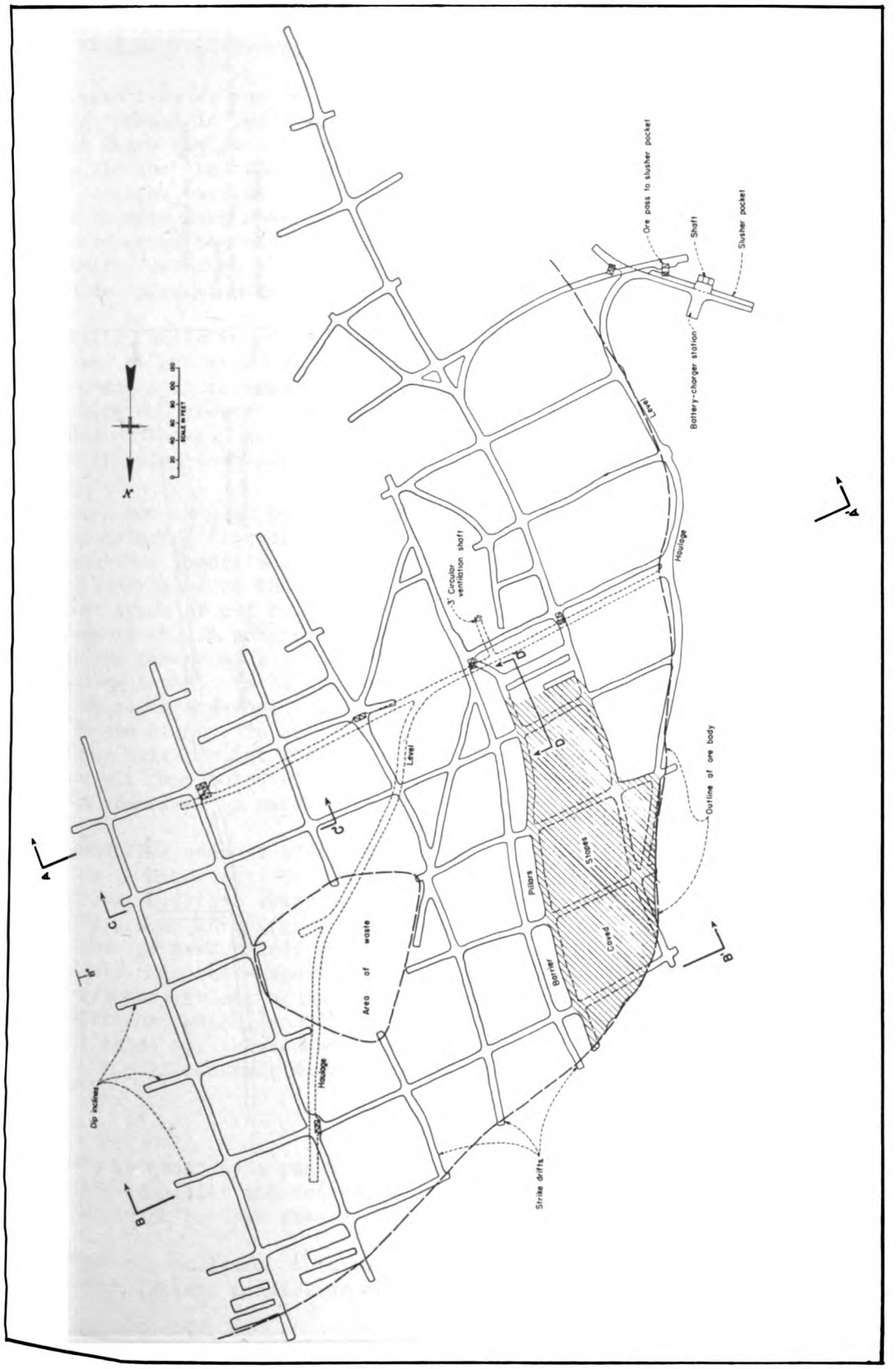

Figure 6. - Plan of underground workings.

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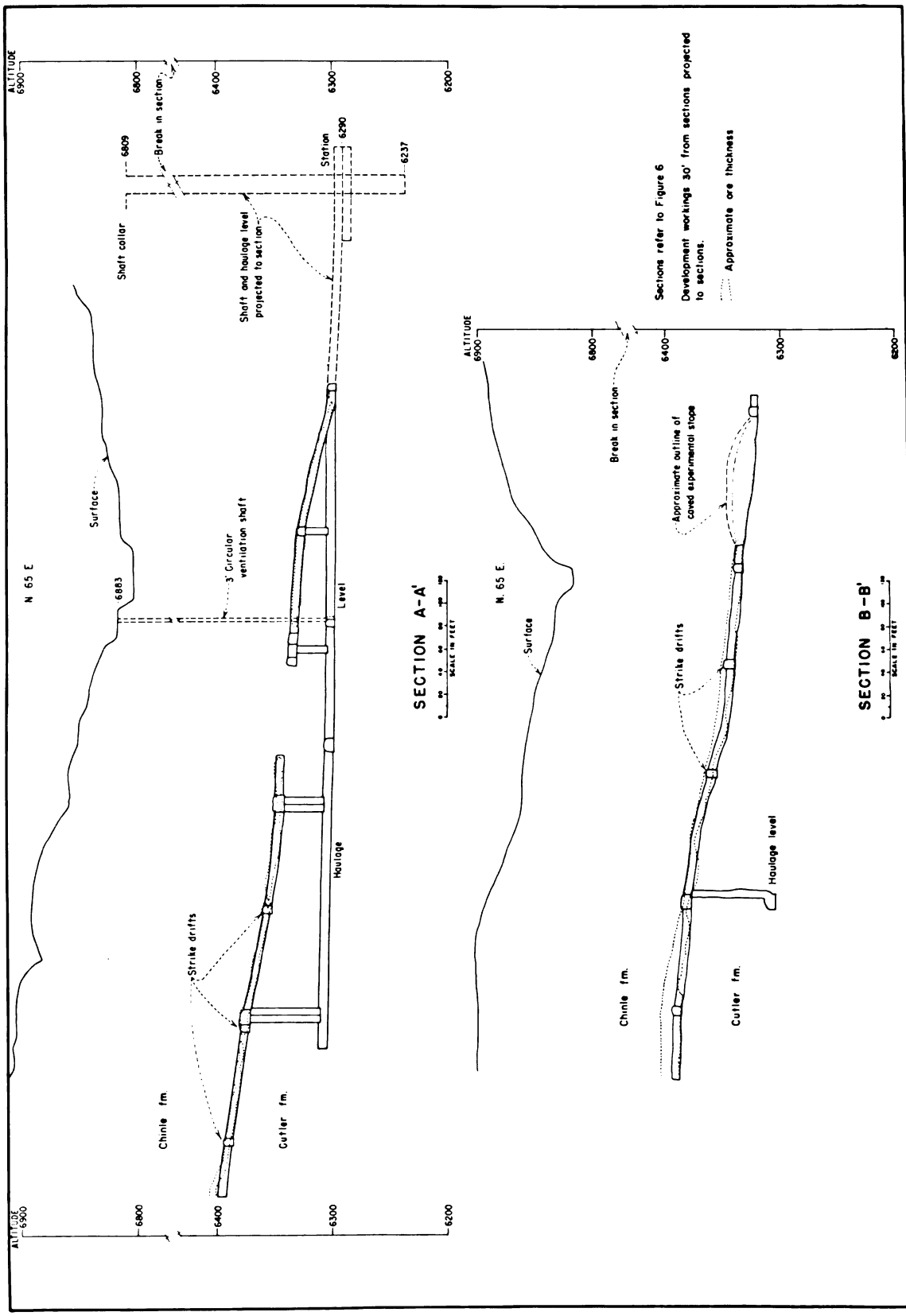

Figure 7. - Sections A-A' and B-B' through La Sal mine workings.

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Compressed air is supplied to the development headings through 3-inch pipe. The company utilizes some aluminum pipe, which is cheaper than steel pipe and easier to handle. Split-ring-type couplings are used.

Development-drift rounds are drilled with 45-pound rock drills, mounted on telescoping, pneumatic legs, which have 6 feet of travel. When drilling in the ore stratum where the rock is predominantly sandstone, good results are obtained from using 7/8-inch hexagonal, 6-foot-long drill steel, with integral 1-5/8-inch-diameter, tungsten carbide, insert-type chisel bits. The drilling rate of the chisel bit is decreased when the bit is used in soft mudstone, because the cutting edges seem to crowd the mudstone and have less chipping action. A 4-wing, tungsten carbide-insert, detachable bit has given the best results in the Cutler formation, where mudstone predominates.

Basically, drift rounds are patterned around a 5-hole burn cut with minor modifications as preferred by the miner or as required by the rock. The number of blastholes necessary to break a 7- by 7- or 7- by 8-foot heading round ranges from 34 to 41, with the greater number of holes and least burden per hole required in tight, solid mudstone. At least 6 trim holes are necessary to break the sides clean when drifting in mudstone. Figure 8 shows two typical drift rounds.

The burn cut is used most commonly for driving standard-size development headings on the Colorado Plateau. Usually it produces the initial opening, regardless of varying ground conditions. Parallel burn-cut holes are easier to drill and require less experience on the part of the driller than is necessary to direct and to bottom other types of cut holes. To produce a clean cut, the cut holes are drilled in sandstone if at all possible. At the La Sal mine the cut holes are not always grouped in the center of a round but often are moved up or down to take advantage of a sandstone lense. If the burn-cut holes are too close or are detonated together when the cut is in mudstone, the rock within the cut may "freeze" rather than be expelled by the blast. To eliminate freezing, the outer four cut holes are shot diagonally in pairs rather than together. Sometimes, the cut-hole pattern is rotated so that all four holes will break across the bedding planes. A 2-1/2-inch center hole sometimes is used in soft mudstone.

The explosive used is 40-percent dynamite in 8-inch cartridges, either 1 or 1-1/8 inches in diameter. The 1-inch-diameter cartridges are used to load rounds drilled in soft mudstone where hole squeezing is a problem. It is common practice to place the primer cartridge at the bottom of the hole. Standard-delay, electric blasting caps are used, wired in series. To facilitate scraper mucking, lifters are shot last. Eighteen-gage annunciator wires are used with Dupont model CD-32 condenser-discharge blasting machines. Heading rounds are detonated from temporary blasting stations, which are 300 to 500 feet from the face. On the stope level, one temporary station may service a number of headings. Blasting is done at the ends of the shifts, but if contract miners are working alone, they often blast at noon.

Mining

The ore is mined by a panel-retreating system. The mining method is a combination of room-and-pillar and retreat mining. The aim is to limit the mining area and to induce the back to cave when extraction has been completed.

Although the attitude, thickness, grade, and uniformity of the La Sal ore body were important factors considered in choosing a mining method, the thickness of the

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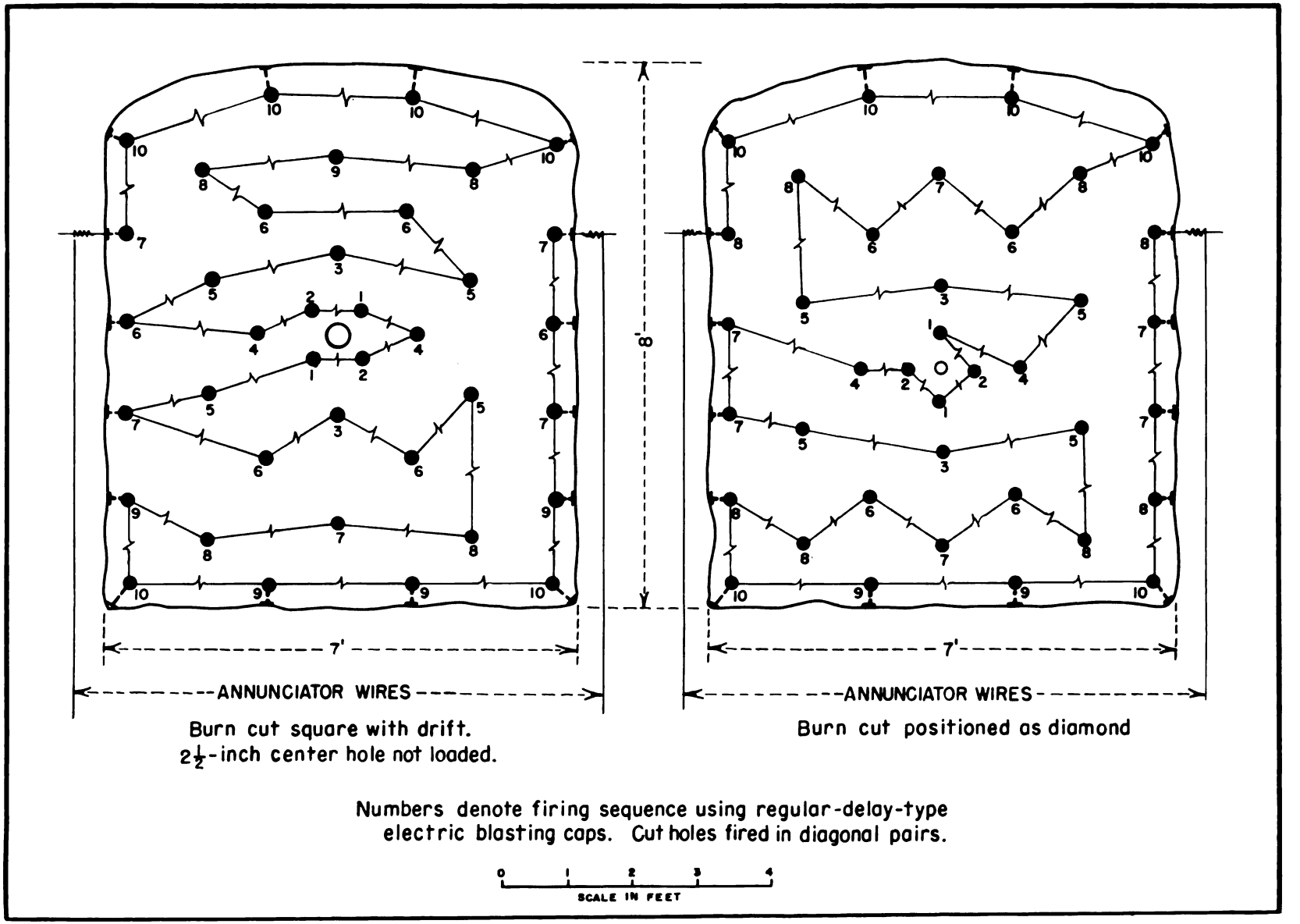
8'
7'
ANNUNCIATOR WIRES
Burn cut square with drift.
2½-inch center hole not loaded.
7'
ANNUNCIATOR WIRES
Burn cut positioned as diamond
Numbers denote firing sequence using regular-delay-type electric blasting caps. Cut holes fired in diagonal pairs.
0 1 2 3 4
SCALE IN FEET

Figure 8. - Two 39-hole drift-round patterns, using 5-hole burn cuts.

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ore and the type of rock overlying the ore were the final determining factors. The high value of the ore makes it essential that the mining method obtain the highest possible ore recovery.

Exploratory drilling and early development revealed the weakness of the rock that caps most of the ore. Because the back rock lacks strength and rigidity, original planning rejected the conventional room-and-pillar system. It was decided that rooms wide enough to permit room-and-pillar mining would be difficult to keep open, pillar recovery would be costly, and loss of ore in pillars would be high. It was considered important to select a mining method that, after mining was begun in a block, would permit rapid extraction to insure maximum recovery of ore and safety of personnel. Dividing the ore body into blocks with a development grid not only defines the ore body but also provides access to the stoping units so that some type of retreat mining can be practiced. The dimensions of the grid were not based on any particular method of retreat.

Before development was completed, experimental retreat stoping was begun in 4 blocks between the 2 lower strike drifts near the northwest boundary of the ore body, which was the only ore limit then established. In March 1956 work was begun to determine what width panel could be mined before caving would begin and to find the most effective method of supporting the backs at the working faces. Various schemes were tried. Twenty-foot barrier pillars were established first along the upper sides of the blocks to protect the upper strike drift. A panel stope, 50 feet in width, was begun at the north end of the north block. The ore was broken across the face and slushed to the dip incline, then down to the lower strike drift. As mining progressed downdip, the working face was protected by timber stulls and 6- and 8-foot roof bolts. Caving began when the stope had progressed about halfway down the block.

The second stope panel was 100 feet in width and extended 50 feet on each side of the second dip incline. The working faces were protected by roof bolts, stulls, and 6-foot-square timber cribs. Caving also began when the faces had been advanced about halfway down the block. As the span was increased, the size of the caving material increased.

A third panel, 50 feet in width, was then begun at the south end of the second block. Here, small chain pillars were left between the new panel and caved area. The face was protected with more closely spaced cribs and with stulls and roof bolts. The small chain pillars were recovered as the face was advanced downdip. Caving started when the stope had been advanced two-thirds of the distance down the block. The unmined ore remaining in the downdip portions of the blocks was mined by working updip from the lower strike drift. Although recovery was good in the first two experimental blocks, the wide stopes proved to be unsuccessful, because caving occurred prematurely. As the span was enlarged it became more difficult to control caving.

An entirely new panel system was tried - and proved successful - in the third and fourth experimental blocks; this new system also has been started in the northeast end of the ore body. The ore is mined in 18-foot panels, which are carried the full length of the block, parallel to the dip of the ore body. Two-thirds of each panel is mined on the advance and one-third on the retreat. The advance section is mined by driving a 12-foot room updip from the lower strike drift, leaving a 6-foot pillar between it and the area previously mined and caved. The room is driven to the limit of the block. The 6-foot section is mined by recovering the pillar between the room and the cave, retreating downdip. An idealized sketch of this mining method is shown in figure 9 and a completed room in figure 10.

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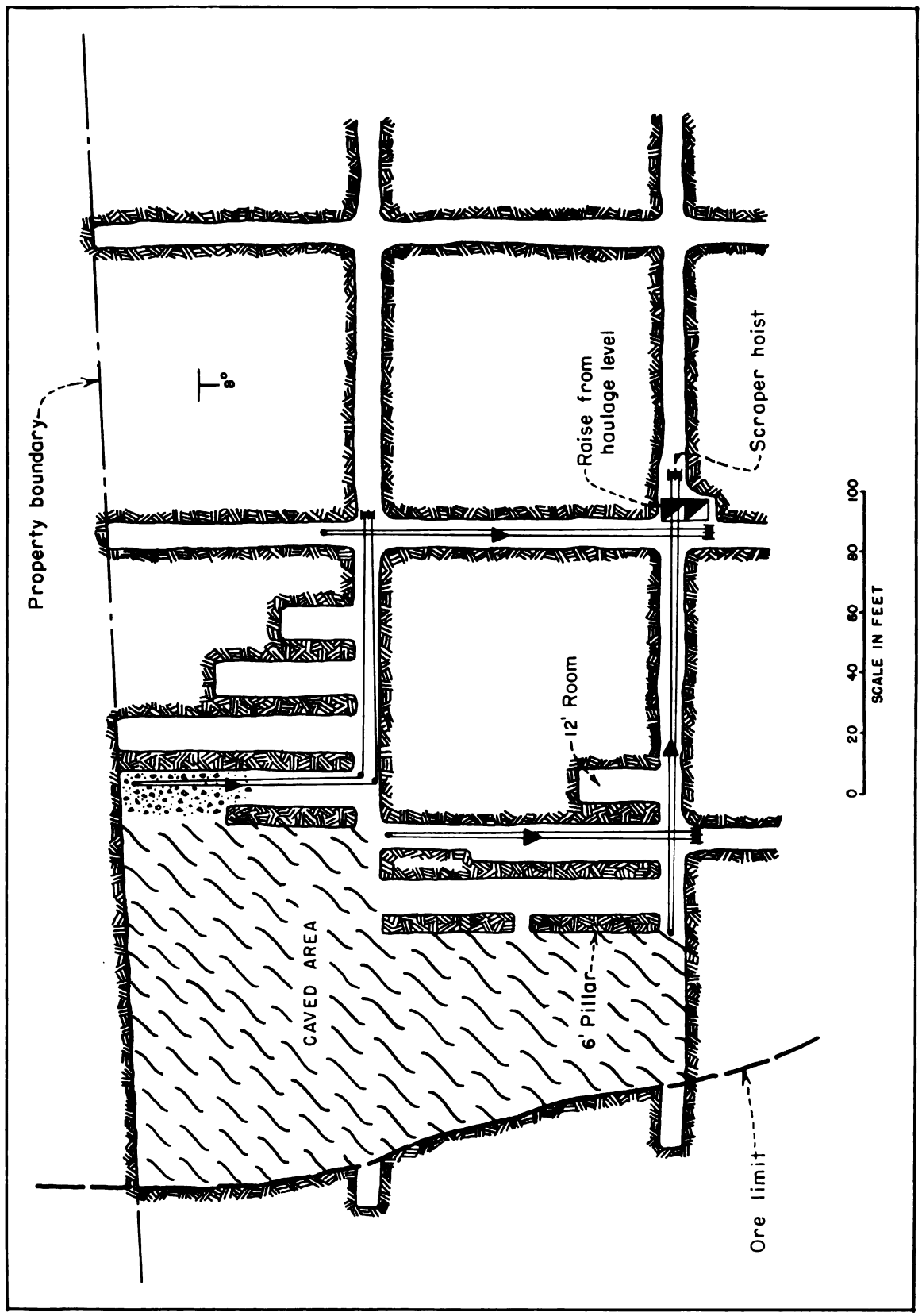

Figure 9. - Idealized sketch of mining method.

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Figure 10. - Barring down in 12-foot-wide room.

Figure 11. - Loading blastholes drilled at 45° with pillar.

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The work is done in cycles, so that three panels are being mined in progressive stages. Three rooms are driven on a staggered line of advance so that the room nearest to the caved area is completed first. The ore is scraped to the lower strike drift, then to an ore pass. When the first room is completed, a fourth is started, keeping its advance behind that of the last. Recovery is then begun on the pillar between the first room and the cave. Working within the room, a line of 6-foot roof bolts is installed on 3- to 4-foot centers along the rib of this pillar. The roof-bolt holes are inclined slightly to bottom inside the ribline. Next a row of vertical cutoff blastholes is drilled into the back along the other side of the room, parallel to and about 1-1/2 feet from the rib of the second pillar. These holes are drilled 6 feet deep on about 2-1/2-foot centers. The pillar to be recovered is then divided into two sections by driving a 4-foot hole through its center to the caved area. Blastholes are drilled across both pillar sections on about 2-foot centers (fig. 11). The holes are pointed at an angle of 45° with the rib and bottomed about 6 inches from the opposite rib. The two pillar sections are blasted separately, beginning with the one farthest updip. The broken ore from the upper section is scraped downdip to the lower strike drift before the lower section is broken. The roof bolts help to prevent the back from sloughing while the pillar is being recovered (figs. 12 and 13). After the second pillar section has been recovered, the vertical cutoff holes along the next pillar are loaded and blasted to induce caving against this pillar. The rock, which is forced to cave, breaks over the top of the roof bolts. The caved material along the pillar and the angled blastholes minimize the amount of ore blasted into the caved area. The back continues to cave and finally arches. One hundred feet inside the caved section from experimental block 4, the back is nearly 50 feet high; the pile of broken material has almost filled the chamber created by the caving. It might be necessary to reduce the width of the panels in areas where the ground is harder to support and to leave a small stub pillar at the downdip end of the lower pillar section while the pillar next to it is being recovered.

This method of panel mining minimizes the amount of roof support required, lends itself to efficient scraper mining, and permits good cycling of the work.

The rooms are driven with a drill round having a horizontal V-cut (fig. 14). In ore 8 to 10 feet in thickness 42 to 50 holes are drilled, and hole spacing varies according to the nature of the ground and the preference of the miners. Two miners drill the V-cut round, each using jackleg-mounted drills. To drive the room straight and keep the pillars at uniform width, centerlines are surveyed on the faces before drilling.

Mining and development costs are summarized at the end of this report.

Underground Transportation

Track Haulage

Track-type haulage equipment is used on the main haulage level and in portions of the first three strike drifts where the haulage distances exceed 200 feet.

A Titan, 2-1/2-ton, battery-powered locomotive is used to haul six 35-cubic-foot-capacity, rocker-dump-type mine cars on the lower level. The track has a 30-inch gage and consists of 30-pound rails, spiked to 6- by 6-inch native-pine ties laid about 3 feet apart. The track on the main haulage level was laid on an approximate 0.5-percent grade. The longest haul is about 1,500 feet.

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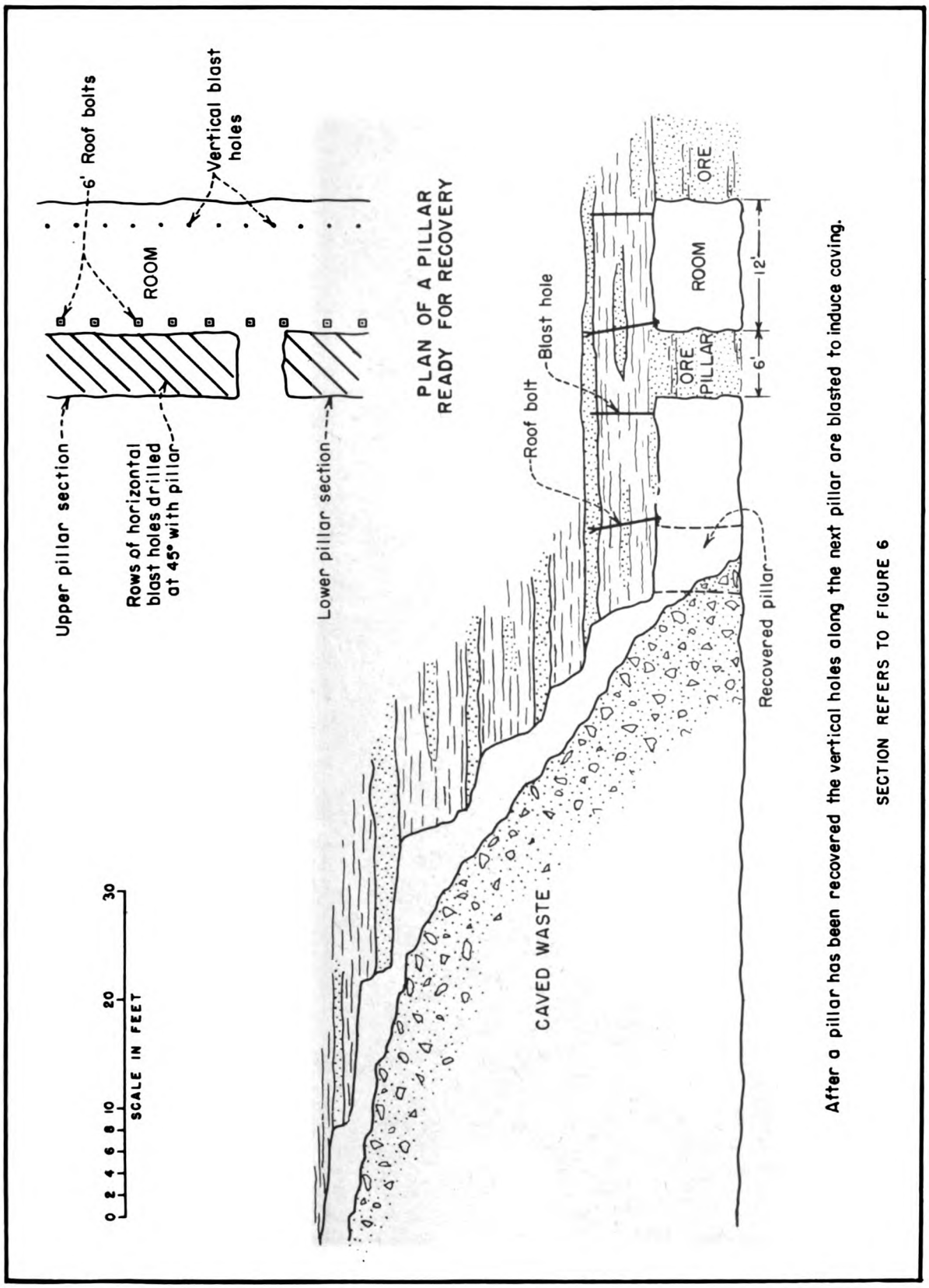

Figure 12. - Vertical section D-D' through stope blocks.

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Figure 13. - View of caved area after pillar recovery.

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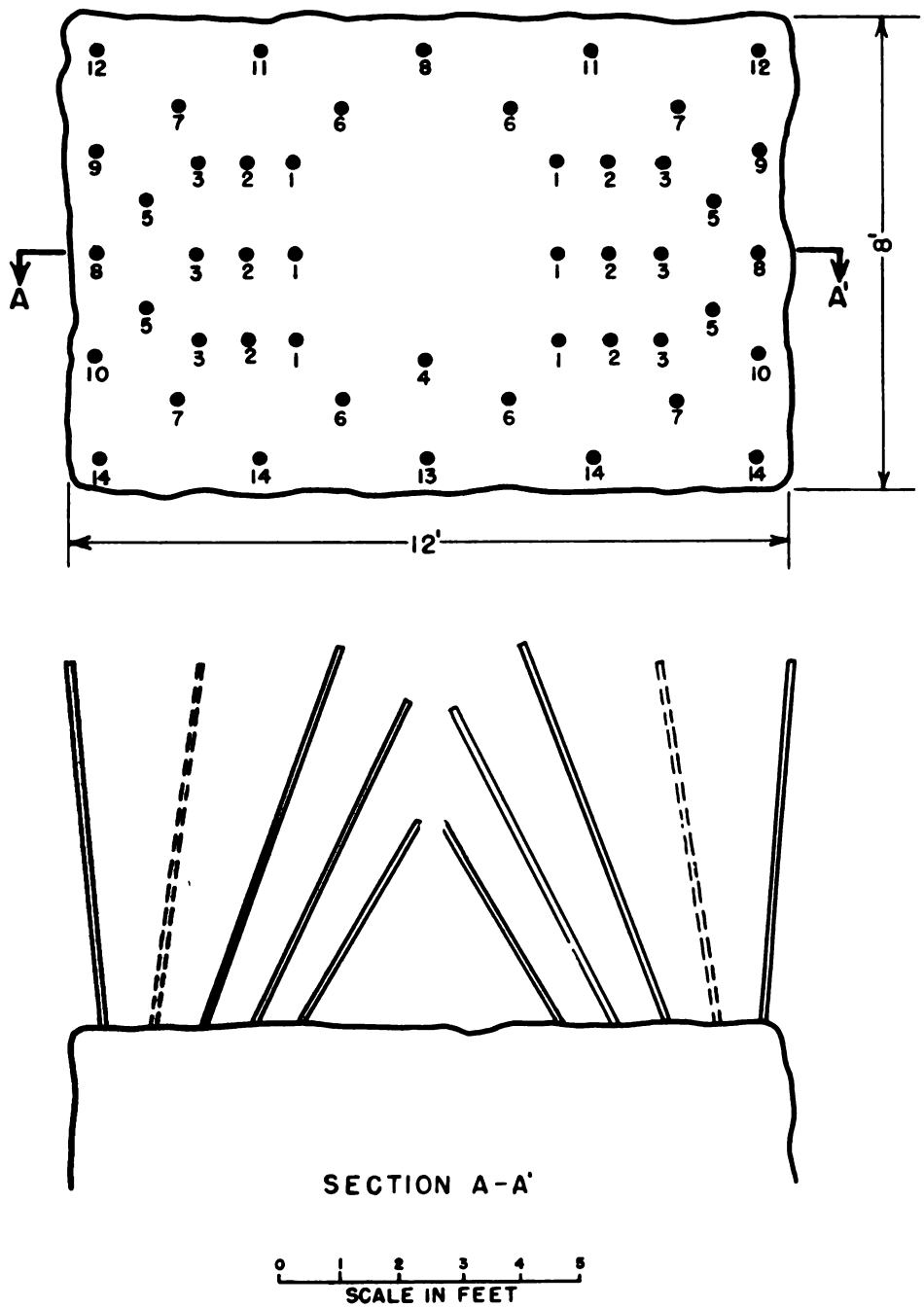

Numbers denote blasting sequence using millisecond-delay-type blasting caps.

Figure 14. - A 47-hole drill round with horizontal V-cut, used to drive 12-foot rooms.

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Five ore passes connect the haulage level with the stoping area. The ore passes are 25 to 65 feet in length and progressively longer farther from the shaft.

Ore chutes are constructed of 8- by 8- and 6- by 6-inch Douglas-fir timber, equipped with hand-operated steel gates. The mouths of the chutes are lined with 3/16-inch Jalloy plate steel and inclined approximately 50°.

When an ore train enters the station area, the cars are positioned alongside the slusher pocket; each is tipped to empty into the pocket without repositioning the train or uncoupling the cars. It takes about 3 minutes to empty a train of 6 cars. Ore is hauled on the lowest strike drift in the same type of mine cars by a Mancha, 1-1/2-ton battery-powered locomotive. The track is the same size as that on the main haulage level. Track-type haulage equipment will be used on this and the next 2 levels where transportation exceeds 200 feet. A crosscut was driven from the lowest strike drift toward the shaft; and a 5- by 6-foot untimbered ore pass, 25 feet in length, connects this level with the slusher pocket on the station. Eimco No. 12-B RockerShovels are used when driving drifts equipped with track. Scraper and scraper slides are used when driving headings from these drifts. Semi-permanent, timber scraper slides are built at the entrance to dip inclines.

Scraper Haulage

The development layout and mining techniques are based mainly on the capacity of scraper mining. Scrapers were chosen because of low headroom and the need of a flexible method of moving ore from the stopes and through the drifts. In operation, scraper transportation is cheap only when the distance is within economical limits. Maximum scraping distance desired at the La Sal mine is 200 feet. Scraper haulage is planned so that downgrades favor loads. Irregularities at the base of the ore, rough bottoms, working around corners, and working infrequently against the general dip of the ore require the use of average size or larger scraper hoists and scrapers.

Scrapers are used both as loading and conveying equipment. Where the scraping distance exceeds 150 feet or where the ore haul is in 2 directions, the ore often is moved to the transfer point by 2 scrapers. Electric, 15- and 30-horsepower, 2-drum scraper hoists are used with 48- and 54-inch semi-hoe-type scrapers to move the ore broken in development drifts (fig. 15). In the stopes the ore is moved to the drifts by 20- and 30-horsepower, 2- and 3-drum, electric hoists with 48- and 54-inch scrapers. Rope speeds of the 20-horsepower hoists range from 250 to 300 feet per minute.

Because eyebolts and wedges, are lost frequently, the company has replaced the commercial types with a flexible-eyebolt device of its own design - one that is simple and cheap to construct and satisfies its particular needs. The flexible eyebolt is made by inserting both ends of a 36-inch length of 1/2-inch wire rope through a 6-inch length of 1-inch pipe, using babbitt metal to hold the rope in place. By increasing the length of the rope the anchor holes can be placed well ahead of the face in firm ground. This type of anchor permits drilling the anchor holes at any convenient angle; and its flexibility allows the sheave to adjust itself to the angle of pull. The shock and bending stresses are absorbed in the short length of rope with less chance of the wedge working loose and the eyebolt pulling out. Anchor holes in development drifts that were driven in soft ground usually are drilled into the back, so that the pull on the return cable is partly absorbed by the rock at the collar of the anchor hole. The estimated cost of this type of eyebolt is $1.10.

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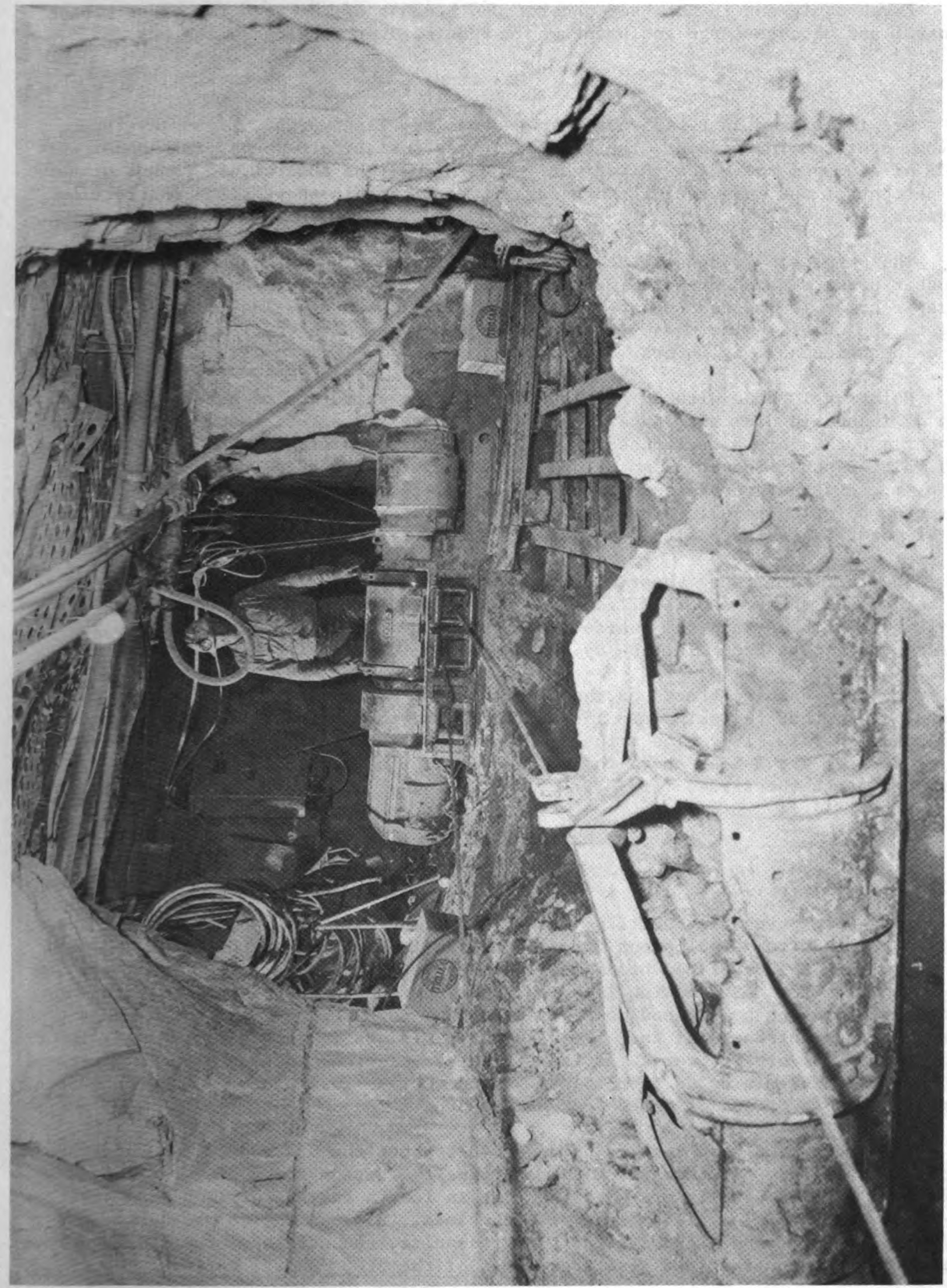

Figure 15. - Scraping ore along strike drift to ore pass.

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Large equipment is not used underground because of the need for close mining control and because large equipment would require overbreaking in low ore.

Hoisting

Ore is hoisted in two 35-cubic-foot-capacity, lightweight, Kimberley-type skips, operated in balance. The crosshead, vertical side members, and bonnet were made of aluminum to reduce the weight. The skip weighs about 1,680 pounds or about 48 percent of the weight of its ore capacity. Figure 16 shows skip-construction details.

The skips are loaded at the station by a scraper, which moves the ore from the slusher pockets to sheet-steel measuring chutes. A 2-drum, 10-horsepower, electric scraper hoist is mounted beneath the station level, and a 42-inch scraper conveys ore to the chutes. The measuring chutes are equipped with vertical gates, operated by 6-inch pneumatic cylinders.

Hoisting is done with an Angelus double-drum, third-motion hoist. The drums are 42 inches in diameter, have 18-1/2-inch faces, and are 51 inches center-to-center. Both drums are clutched and have postoperated brakes. An 80-horsepower, 440-volt, 3-phase induction motor drives the hoist. The speed at the drum is 50 r. p. m., and the average hoisting speed is 520 feet per minute. The 3/4-inch-diameter hoisting rope is 6 by 16 improved plow steel, with a hemp center. The hoist motor is equipped with a thrustor-type safety brake.

The headframe is a 4-post Douglas-fir structure, 42 feet in height to the center of the sheave wheels (fig. 17). The 4 main front and back posts are 12- by 12-inch timbers, and the inside front posts are 10- by 10-inch timbers. The braces and struts are mainly 8- by 8- and 10- by 10-inch timbers.

The skip-dumping tracks on the headframe are a standard Kimberley-type design, installed so that the skips are 15-1/2 feet above the shaft collar when in a dumping position (fig. 18). The skips dump into short sheet-steel slides, the bottom lips of which are 9 feet above the shaft collar. The ore falls into a dump truck parked beneath the slides (fig. 19).

Auxiliary Operations

Long-Hole Drilling

Long-hole percussion drilling has an important part in mine development. Long holes, drilled from development drifts, furnish information on thickness and grade of ore within the developed blocks and help to establish the minable limits of the deposit (fig. 20). By knowing in advance the average height of the ore, stoping within a block can be planned more efficiently. Advance knowledge of an erratic change in the thickness of the ore or in the divisions in the ore stratum is important to maximum ore recovery. At the Little Beaver mine, another property operated by the Homestake-Mining Co., ore occurs in the Chinle formation in 2 separate beds, with as much as 6 feet of waste rock between them.

As development headings are driven to block out the ore body, long holes are drilled at angles into each block to determine the bottom and top of the ore. These drill holes usually are 40 to 50 feet in length and are drilled from stations 30 to 50 feet apart on all 4 sides of each block. Because the Chinle-Cutler contact is irregular and often difficult to recognize, the development drifts are sometimes above the contact and also above some of the ore. Short, vertical holes,

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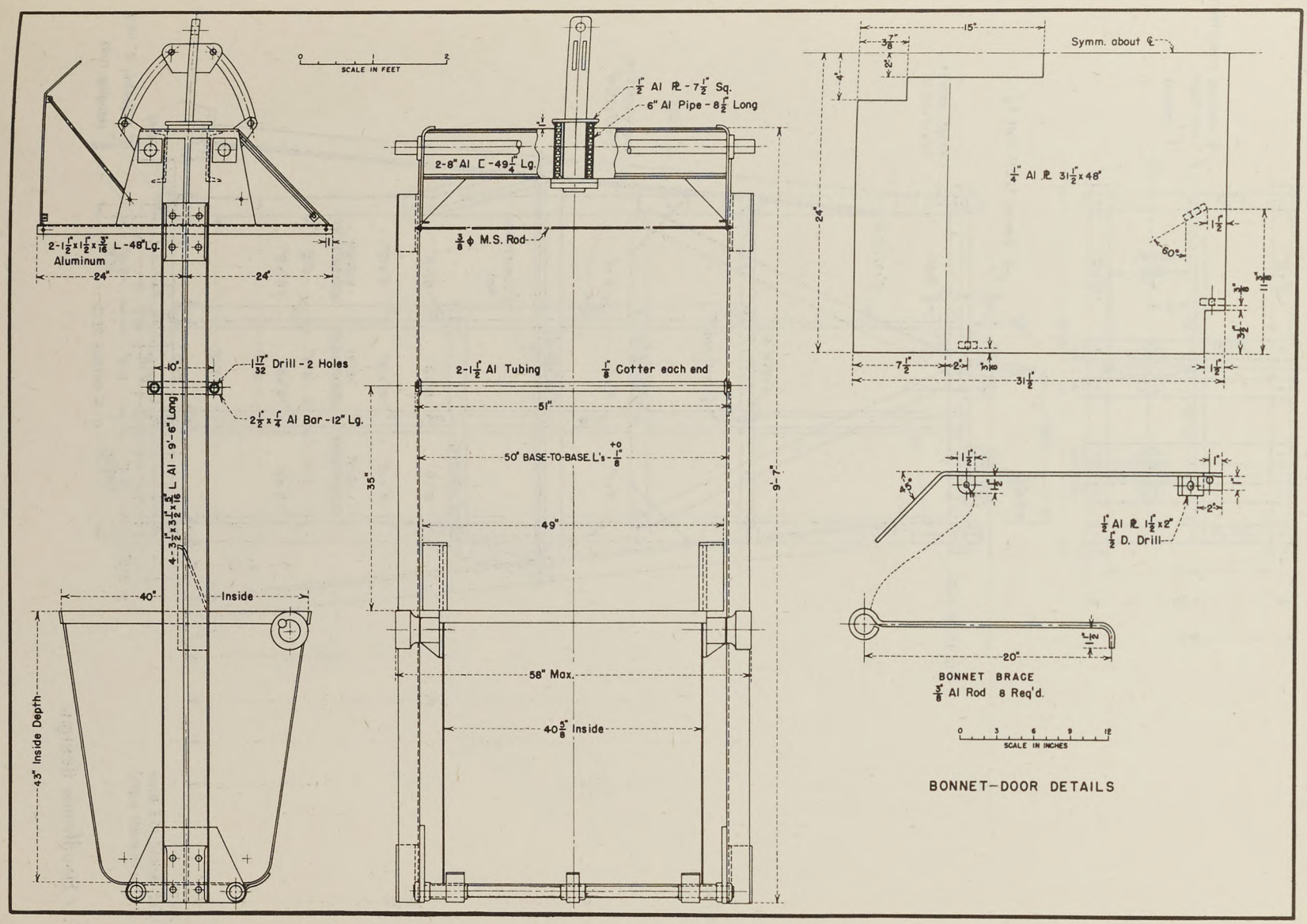

Figure 16. - Lightweight, 35-cubic foot-capacity skip.

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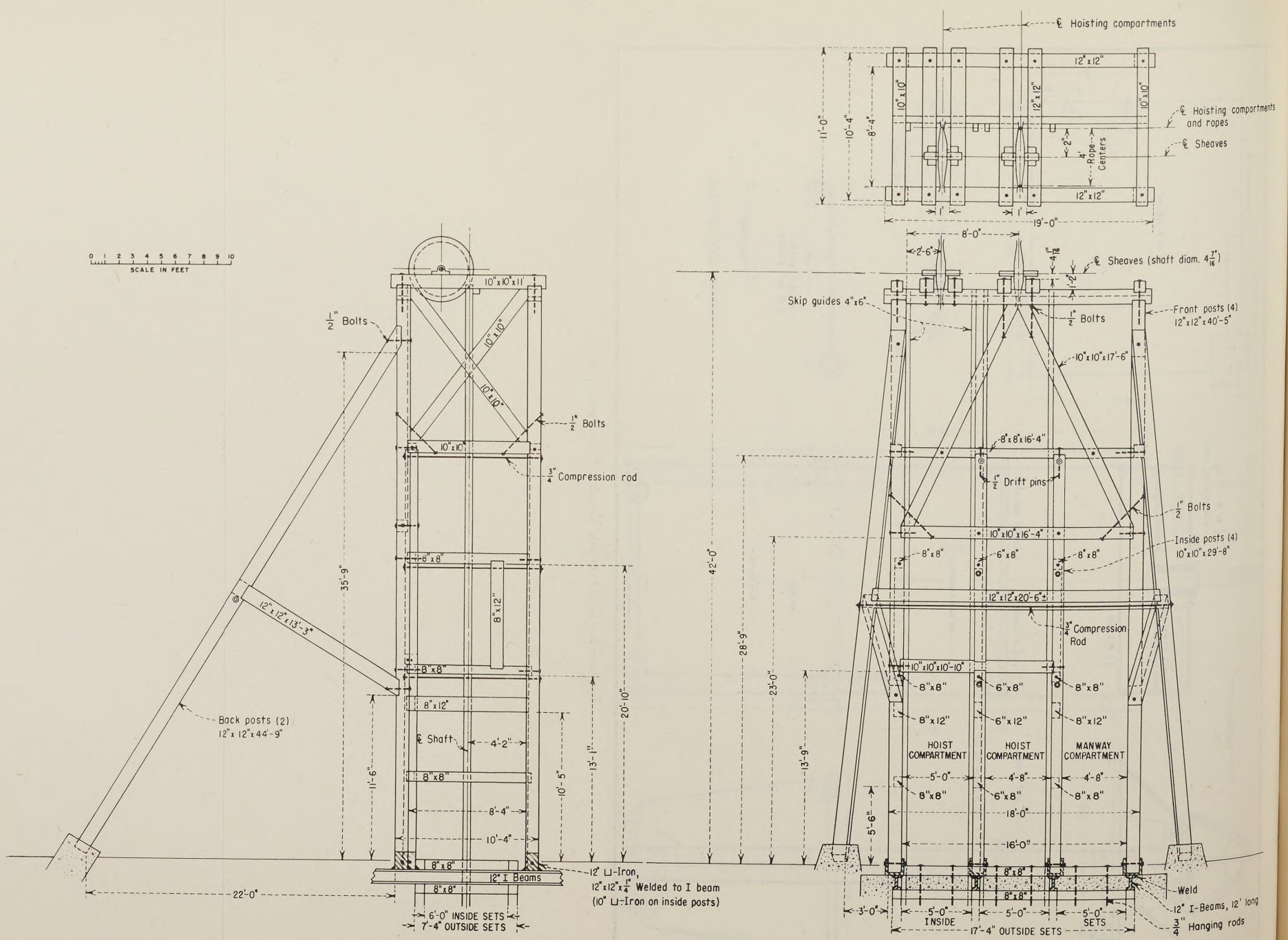

Figure 17. - Headframe design.

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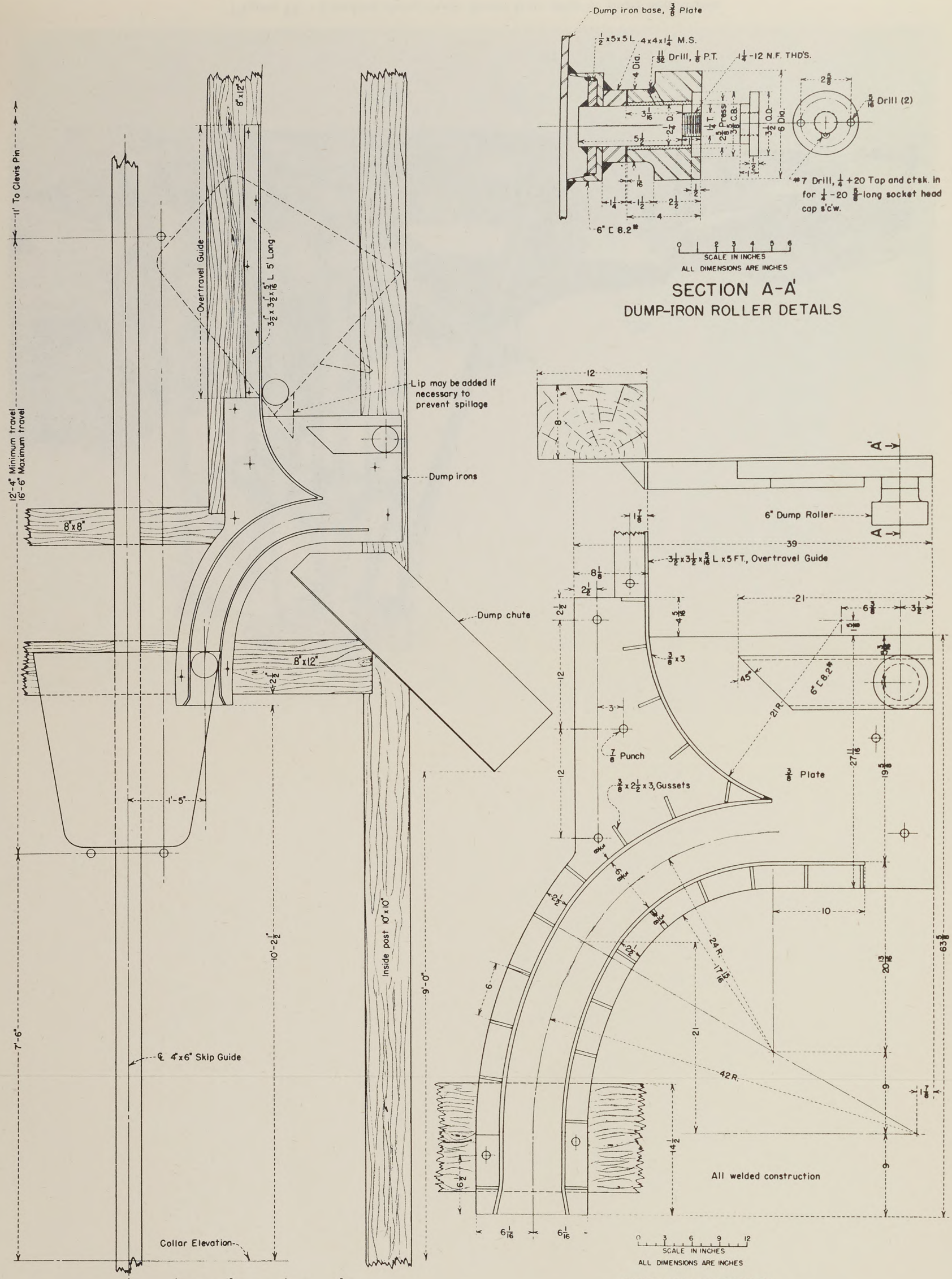

Figure 18. - Dump-iron installation and details.

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Figure 19. - Loading dump truck direct from skip at shaft headframe.

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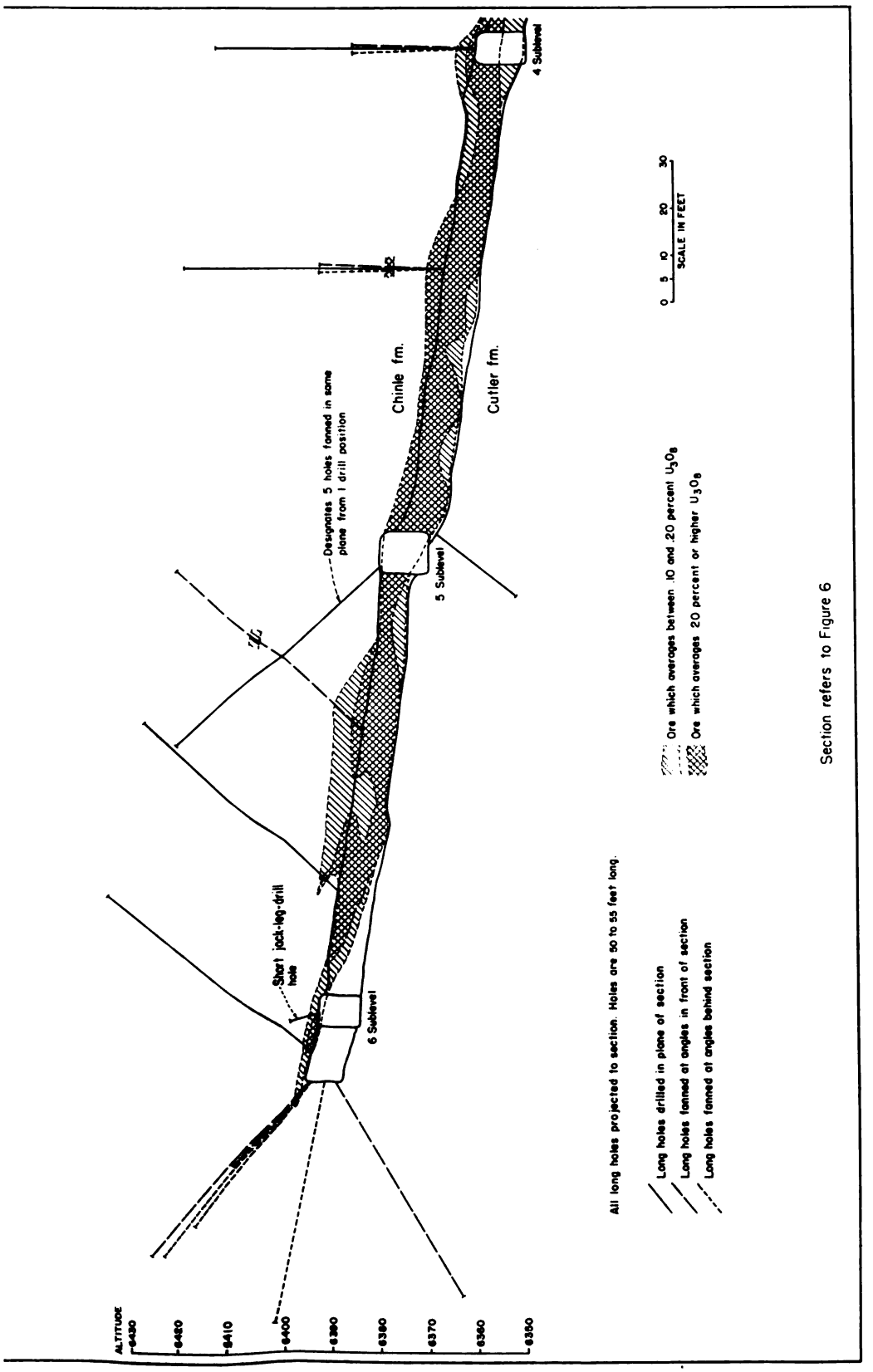

Figure 20. - Vertical section C-C', showing pattern of long-hole drilling.

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drilled downward, have found as much as 2 feet of ore beneath a drift, and upholes as much as 14 feet of ore above a drift.

Long-hole drilling is also used when searching for possible ore extensions beyond the inferred limits of the ore body or in an area where the ore body is not uniform. When a heading was stopped because it was believed to be beyond the known limit of the ore body, long holes drilled from the face, parallel to the bedding, have on occasion indicated additional ore that warranted further drifting.

Although the color of drill cuttings immediately indicates where the bit is penetrating ore or waste, the holes are always checked radiometrically with a long-hole probe and counter. Cuttings are not collected for assaying, except occasionally to check probing results. The probe is calibrated underground at a check hole, which was drilled especially for this purpose and from which the cuttings were carefully saved and assayed chemically. Long holes are plotted on a long-hole assay map to show direction, inclination, and probe-assay results. To date, for every foot of development drifting within the ore stratum, about 1.3 feet of long-hole drilling has been done.

The percussion drill preferred for long-hole drilling has a 4-inch bore and an automatic feed and is mounted on a 6-foot aluminum shell. A rotation-shift lever permits reversing the direction of rotating the steel, which helps when connecting or disconnecting steel. An automatic air-line oiler is used. Drilling is done from a 3-inch-diameter column and crossbar. One-inch hexagonal drill steel in 4-foot lengths is used with 2-1/8-inch, tungsten carbide, insert-type detachable bits. The bit preferred for long-hole drilling has 4 wings and 4 waterways. It has good body clearance and is designed so that the inserts extend 1/8 inch from the body. Its performance in the soft rock is good for about 1,260 feet of hole.

Two men comprise the long-hole drilling crew. They are paid at the rate of $0.55 per foot of hole, which includes payment for time spent in moving, setting up, and making minor repairs. They drill an average of 100 to 110 feet per day.

Roof Bolting

Roof bolts are used to hold the backs at the La Sal mine and have helped to solve the problem of holding the incompetent mudstone in the development drifts and to provide temporary back support in the stopes. Overlying the ore sandstone are irregular lenses of soft mudstone and siltstone of varying thicknesses, which must be secured from sloughing in the development headings and temporarily reinforced within the stopes. Holding an incompetent mudstone is also a problem on much of the haulage level, which was driven principally within mudstone of the Cutler formation.

In development headings the problem is not one of increasing the beam strength of the mudstone across the short span but of controlling sloughing, thereby decreasing ore dilution and the cost of drift maintenance. The material that first falls from the back is that fractured by blasting when the ground is first opened. Sloughing continues after exposure to air causes the mudstone to dry and part from the solid material behind it. After a natural arch is formed at the back, the rate of fall decreases; however, it does not always stop. Sloughing in a few instances has continued to stope upward until the drift is almost twice its original height. Standard size wood or steel bearing plates are used where the back is sandstone. This type of bearing plate is of little value where the back is mudstone. The mudstone at first sloughs from around the plate and then from behind it and finally upward around the bolt. To increase the bearing area around each bolt, the company first

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tried bolting punched aircraft landing-mats (war-surplus) to the roof. Although this method increases the area covered per bolt, the mats are expensive and not always available. Sloughing sometimes occurred around and behind the mats much in the same manner as it did near the smaller bearing plates. The company then experimented with heavy, chain-link, wire mesh. This method has proved most successful, because greater area can be covered in one operation; greater coverage is provided per bolt, and the flexibility of the wire mesh permits fitting it to the irregular contours along the back of a drift (fig. 21). Material that breaks loose is held from falling by the wire mesh, and the rate of spalling is progressively slowed behind the broken material. The wire is strong enough to support large pieces of rock.

Nine-gage, ungalvanized-steel, chain-link, wire mesh (purchased in 50-foot rolls in 42- and 60-inch widths) is bolted to the back of development drifts by 1-inch, round, 4- and 6-foot-long, mild-steel, slit-rod-and-wedge-type bolts. Triangular or square mild-steel bearing plates, 8 inches on the side and 3/8 inch in thickness, are used.

In the first two experimental stopes, roof bolts were used in conjunction with timber to hold the rock strata over wide spans. Roof bolts, 6 and 8 feet long, were installed on 5- to 10-foot centers. It was learned that the mudstone could not be depended upon to act as a rigid beam across wide spans. The mudstone fell in progressively larger pieces as the span was increased, parting at the top of the bolts. The mudstone could not be held by securing the roof bolts in more substantial overlying rock strata, because such strata were not always at reachable heights. Thin sandstone lenses above the stope would part at the next mudstone seam. In the 18-foot panels roof bolts are used to minimize sloughing while the 6-foot pillars are covered.

Surface Ore Storage

Ore is stored on the surface in open, bunker-type bins, which are below the level of the mine yard at the bottom of the mine-yard fill. They consist of a timber revetment to hold back the fill and wooden partitions that keep the ore in separate piles, from which it is loaded into trucks and hauled to the mills.

This type of ore storage is favored at the La Sal mine for a number of reasons. It provides larger storage capacity, which was advantageous during the first winter when bad road conditions delayed trucking service, it eliminated freezing of the ore in bins during winter, and exposes the ore for drying in the arid summer. Also, ore mined under contract from adjacent property can be separated from company-mined ore.

The hoisting skips are dumped directly into a 6-ton-capacity dump truck, which shuttles between the shaft and the surface ore bins (fig. 19). The trucks that haul the ore to the mill are loaded with a front-end-type loader. Ore from the La Sal mine is sold to a Government-owned mill at Monticello, Utah. The trucking cost for the 47-mile haul to the mill is $0.0515 per ton-mile.

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Figure 21. - Roof bolting wire mesh to back of strike drift.

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Summary Total of Equipment, Power, and Explosives

Mining equipment used at the La Sal mine consists of the following items:

Number	Equipment item
1	Tractor, crawler, front-end loader, Allis-Chalmers HD-5G.
1	Compressor, 550-c. f. m., Joy WN102, 100-hp. electric motor.
1	Compressor, 525-c. f. m., Gardner-Denver WBJ, 100-hp. electric motor.
1	Compressor, 400-c. f. m., Gardner-Denver WBK, 75-hp. electric motor.
1	Generator, 175-kw., continuous, General Motors.
1	Mine hoist, double-drum Angelus, 75-hp. electric motor.
2	Skips, 35-cu.-ft., Kimberley-type.
1	Fan, 30,000-c. f. m., Joy 36-26 1/2-1750, 1000 series, single-stage, 30-hp. electric motor.
3	Blowers, 5-hp., Coppus TM6.
1	Mine locomotive, storage-battery, 2-1/2-ton, Titan BN.
2	Mine locomotive, storage-battery, 1-1/2-ton, Mancha.
18	Mine cars, 35-cu.-ft., rocker-dump, Couer d'Alene Hardware.
3	Loader, overhead-type, Eimco 12B, RockerShovel.
1	Sump pump, Ingersoll-Rand 35.
1	Pump, high-pressure, 3 by 5, Meyers.
1	Drifter, 4-inch, deep-hole, Gardner-Denver SFH99.
13	Rock drill, airleg-mounted, Cleveland H10AL, with AL-92 telescopic leg.
2	Stoper, Gardner-Denver R-104.
1	Stoper, Gardner-Denver R-94.
1	Scraper hoist, 30-hp., electric, 2-drum, Vulcan DSQ.
2	Scraper hoist, 20-hp., electric, 2-drum, Joy A2F-211.
2	Scraper hoist, 15-hp., electric, 2-drum, Joy A2F-211.
1	Scraper hoist, 20-hp., electric, 3-drum, Joy A2F-311.
2	Scraper hoist, 10-hp., electric, 2-drum, Joy AAF-212.
1	Scraper hoist, 10-hp., electric, 3-drum, Joy FF-311.
1	Scraper hoist, 20-hp., electric, 2-drum, Ingersoll-Rand 20ML2D.
1	Scraper hoist, 5-hp., air, 2-drum, Joy S-211.
4	Tugger hoist, 5-hp., 1-drum, Joy.
1	Tugger hoist, 1-drum, Joy AW-80.

Total power consumption from October 1956 through February 1957 averaged 13.29 kilowatt-hours per ton of ore and waste hoisted. Dynamite consumption during the same period averaged 4.04 pounds per ton of ore produced from development headings and 2.67 pounds per ton of ore produced from the stopes.

SURFACE PLANT

The surface plant consists of four small Stran-steel buildings, built of lightweight steel frames covered by corrugated-steel sheeting. The 20- by 100-foot mine-office building is partitioned into a warehouse, modern dry rooms, mine office, and engineering offices. The 16- by 24-foot shop building is equipped so that most machinery repairs can be made at the mine shop. The 20- by 40-foot compressor house contains 3 compressors and a generator plant. The 16- by 32-foot hoisthouse contains the mine hoist and the lamproom.

Compressed air is supplied by a Joy WN-102, 550-c. f. m., 2-stage compressor and a Gardner-Denver WBJ, 525-c. f. m., 2-stage compressor, each driven by a 100-horsepower, 440-volt, 3-phase induction motor. A Gardner-Denver WBK 400-c. f. m., 2-stage compressor, with a 75-horsepower motor, is maintained for standby operation (fig. 22).

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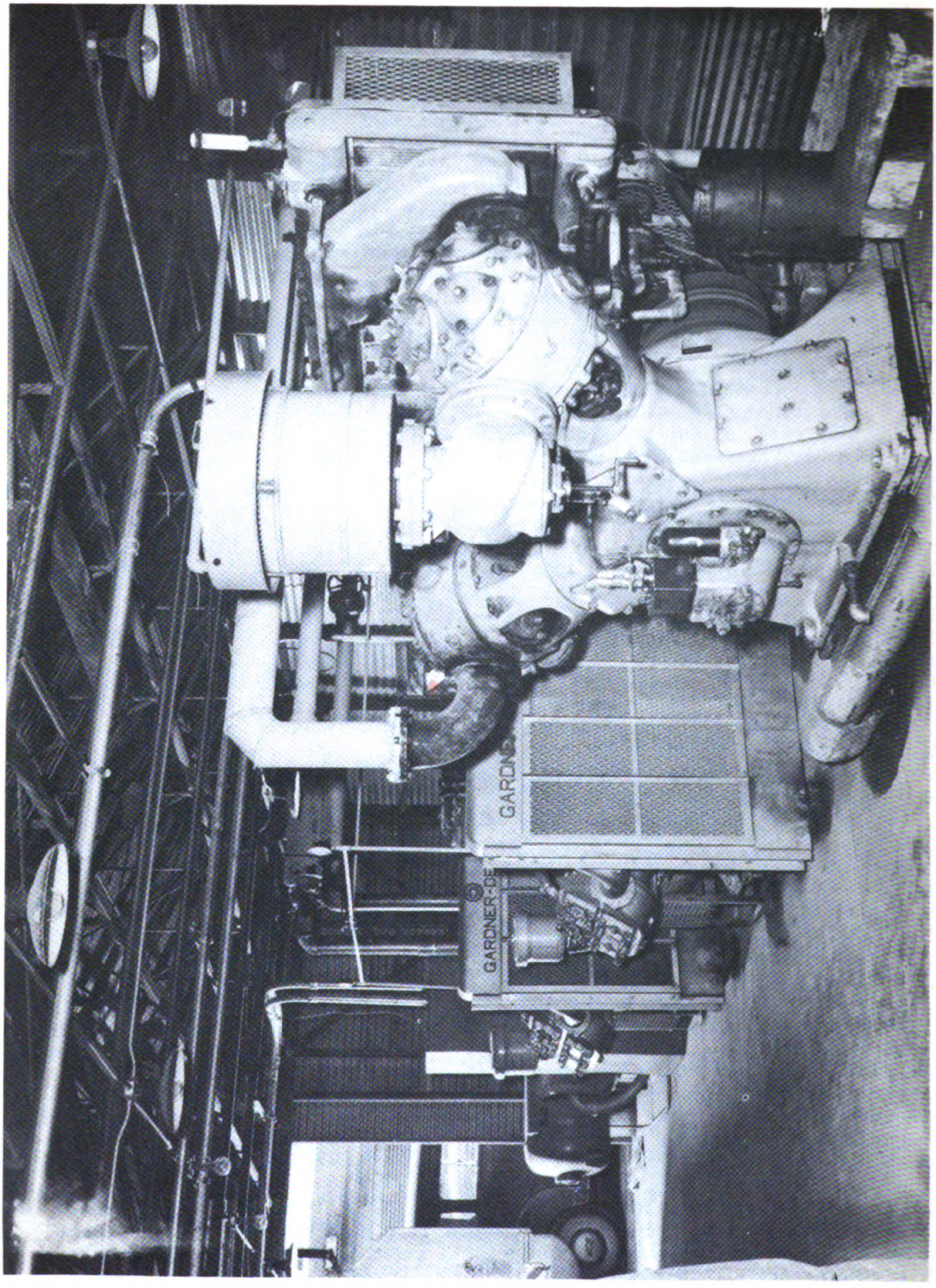

Figure 22. - Compressor building.

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A General Motors diesel-driven generator unit, installed in the compressor house as a standby source of power, supplied electric power at the mine during the early stages of mine development. The generator has a continuous rating of 175 kilowatts and an intermittent rating of 200 kilowatts. Electric power is now furnished by the Utah Power & Light Co. through a transformer station about 2 miles northeast of the mine. A 500-kv.-a. transformer substation at the mine plant steps down the incoming 12,470 volts to 440. The principal mine equipment is operated by 440-volt motors.

Dynamite is purchased in carload lots and stored in the main explosives magazine, which is 500 feet northeast of the shaft in a small canyon isolated from the area of activity. The 10-foot-square magazine is constructed of heavy timbers in a cut excavated in the canyon wall; its top is covered with a thick mantle of soil, and the magazine is well ventilated and dry. It is kept locked at all times, and danger signs are displayed prominently.

Water for mining is hauled in a 1,000-gallon tank truck from a well near the Big Indian copper mine, about 1 mile northeast of the La Sal mine. Water is piped to the buildings and underground from a 3,000-gallon steel storage tank, in an insulated building above the mine yard. A sliding-float indicator on the outside of the building shows the height of the water in the tank.

An Allis-Chalmers crawler tractor, with a front-end-type loader, is used for general maintenance and cleanup work at the mine yard.

Telephone service is maintained through the Moab telephone office by means of a radio-telephone system, which at times operates poorly. An intercommunication telephone line connects the underground shaft station with the hoisthouse and the warehouse.

PRODUCTION RATES, PERCENTAGE OF RECOVERY, SAMPLING, AND ORE-GRADE CONTROLS

Production Rates and Percentage of Recovery

When ore was produced only from development headings, production was 2,000 to 3,200 tons per month. Since stoping was begun, production has steadily increased to 7,500 tons per month. Production of ore from development headings from October 1956 through February 1957, by labor engaged in drilling, blasting, roof bolting, and transporting ore to transfer points averaged 11 tons per man-shift. Production of ore from stopes by labor engaged in the same occupations averaged 16 tons per man-shift. The average rate of production for all underground labor and supervision, including hoisting, was 8.7 tons per man-shift.

No accurate figures are available on percentage of ore recovery using the 18-foot-panel mining method. Except for ore lost in the 3 small unrecovered pillars, extraction from the 4 experimental blocks has been complete.

Sampling and Ore-Grade Control

Much attention is given to sampling and ore-grade control at the La Sal mine. Each working face is sampled daily, and the results are recorded on individual assay maps of the mine workings. After considerable experience, the technique of radiometric sampling has been improved, making it possible to sample a face quite accurately by scanning it with a geiger counter. Sample scanning is quick and gives results that can be interpreted and used immediately. Assay-control maps, kept current with the work completed each day, provide up-to-date records of ore-grade changes.

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To sample underground workings properly by scanning, it is necessary to eliminate as much background effect as possible and to adjust for background changes that cannot be eliminated entirely and for changes in the sensitivity of the counter. A lead shield on the probe reduces radiation other than emissions from the point on the face being sampled. To correct for changes in the background count and in the sensitivity of the counter, the counter is calibrated underground immediately before scanning by placing the probe over a channel from which the cuttings were assayed chemically. With the probe placed on the cut, the calibration knob is turned so that the needle of the meter indicates an amount equal to the U_3O_8 value of the cut. The calibration cut is in a development drift a short distance from the shaft. It is marked clearly so that the probe always can be placed in the same position over the cut. By calibrating the counter in this manner, it is possible to adjust for considerable background effect.

The judgment and experience of the sampler are of paramount importance. He must know the characteristics of the counter and the nature of the ore to be able to interpret the counter readings properly. As the probe is moved vertically across a face, the sampler must estimate the average U_3O_8 content by noting the mean reading in the swing of the meter needle (fig. 23). Because the ore occurs so erratically in strata crossing the face, it is almost impossible to estimate accurately the average grade of the ore when scanning a face from top to bottom. For best results, the scanner divides the face into short, vertical sections across which the grade is nearly uniform. He then estimates the grade across each section and records the average values and widths of the samples on the drift-assay map. A typical face might show the following data on width and grade, beginning at the back and proceeding downward:

	U_3O_8, percent
First 6 inches	Trace.
Next 2 feet	0.25
Next 6 inches	.80
Next 2 feet	.34
Next 1 foot	.12
Next 6 inches	No count.
Cutler formation	

For convenience, samples are taken at intervals of 1 foot or more. Changes in color help to determine intervals of change in grade. Counter readings over 1 percent U_3O_8 are recorded as 1 percent plus, because the accuracy of the counter is questionable when higher grade ores are being scanned. A typical drift-assay map is shown in figure 24.

Ore-grade control is watched closely in development drifts. Faces of development headings are checked after every round, and the results are logged. Development scanning is vital where the bottom is very irregular and the drift must be raised or lowered to follow vertical changes; it is also important when approaching boundaries of the ore body or when the grade of the rock is questionable. Sample scanning during the development stage provides valuable information on grade of ore for the stope blocks. Daily determinations, as recorded on assay maps, are helpful in ore-waste determinations, ore blending, and periodic grade calculations. To check the results of scanning, chip samples are taken periodically from the working faces that are scanned and radiometrically assayed on a nuclear scaler. Radiometric scanning is favored at the La Sal mine, because in the time needed to cut a sample and crush, pulverize, and count it on the scaler, an experienced scanner can scan 3 or 4 faces.

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Figure 23. - Radiometric sample scanning.

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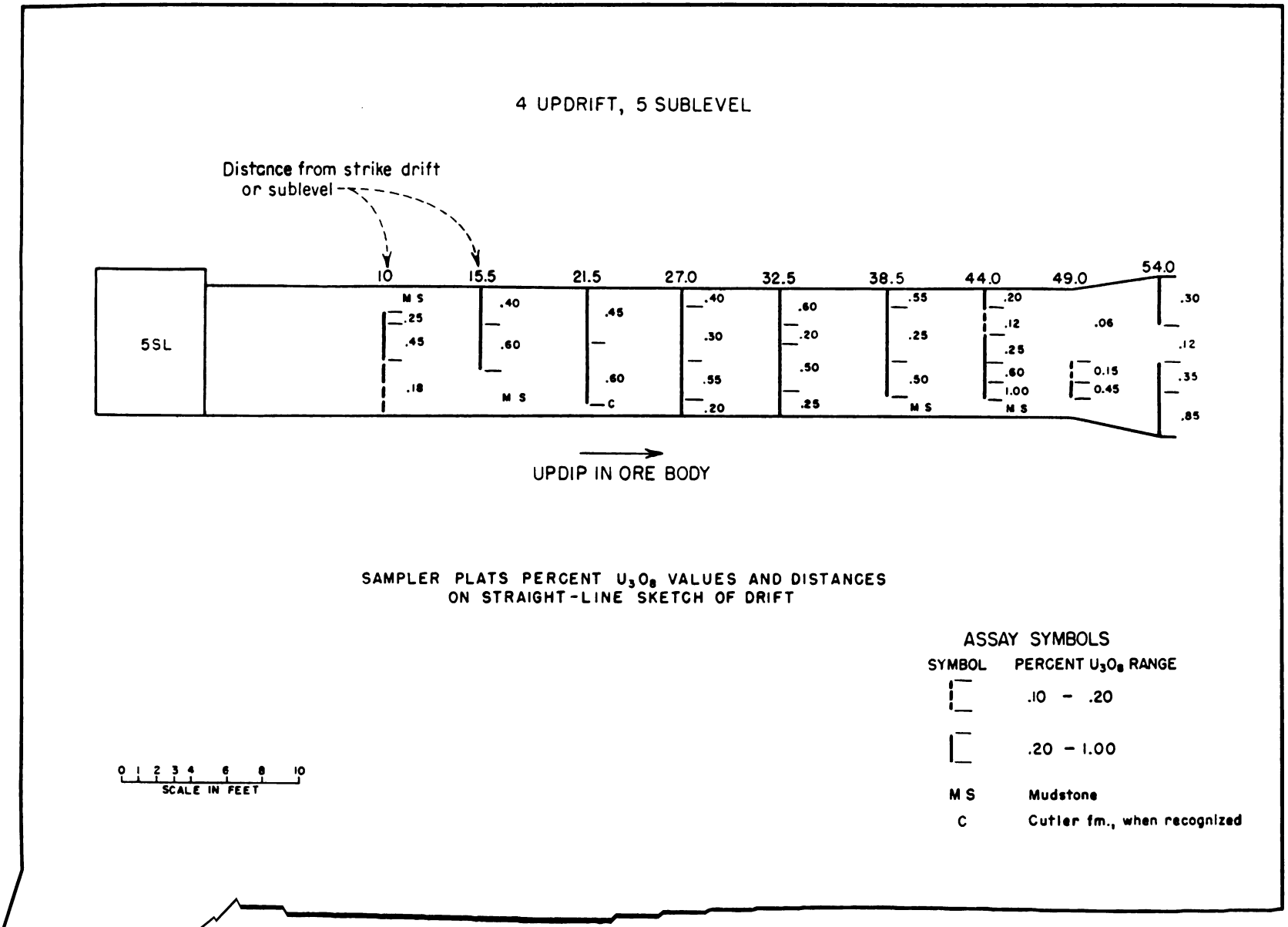

Figure 24. - Typical radiometric-assay sample section through a development drift.

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The grade of daily production is estimated by grab sampling: Samples of ore trammed from each heading and stope are collected from mine cars at the shaft station, then pulverized to about minus-60-to-80 mesh in a 4-inch jaw crusher and radiometrically assayed on a scaler for their U_3O_8 content, which is compared with the grade reported on mill-settlement sheets. Pulp samples returned from the mill (representing 250-ton lots) also are checked on the scaler and occasionally sent out to be analyzed chemically for U_3O_8, V_2O_5, and $CaCO_3$ content.

MINING METHODS IN DISTRICT

Nine underground mines are operating in the Chinle formation along a 5-mile strip of the escarpment bordering the southwest side of Big Indian Wash. Including the La Sal mine, 5 are developed through shafts, 2 through adits, and 2 through slope-type entries. The other 4 shaft mines and 1 of the slope mines are in various stages of underground development. Two nonoperating shaft mines are in the immediate area, which also are in the Chinle formation. Chinle ore is being produced from a slope-developed mine 10-1/2 miles southeast of the La Sal, and ore from the Salt Wash member of the Morrison formation is being mined from an opencut about 5-1/2 miles northwest of the La Sal mine.

Usually vertical shafts are sunk near the lowest ore within the boundaries of the property. Development work is done both within and beneath the ore. In some mines the ore is developed by a system of nearly parallel drifts, usually driven along the strike of the ore body; in others, by a system of dip inclines. Open stoping is the principal mining method. One company is planning to use a variation of longwall mining, with collapsible steel props to support the back near the working faces. A few mines are using standard room-and-pillar systems and will recover pillars on the retreat.

The thickest Chinle ore found so far in this district is in the Mi Vida and Big Buck mines. Generally, the ore thins northwest of the Mi Vida, Big Buck, and Little Beaver mines, and the rock overlying the ore becomes progressively more difficult to support. The problem of mining thinner ore and the more acute problem of supporting the back have forced operators at the northwestern end of the area to try various schemes based on retreat mining rather than on the conventional room-and-pillar method.

VENTILATION

The La Sal mine is ventilated by mechanical means. Fresh air is forced through mine workings by a controlled ventilation system, with the principal aim of reducing the radioactive concentration of daughter elements of radon in the mine atmosphere. If enough fresh air is supplied at the working places to dilute and remove airborne radioactive substances, the fresh-air requirements will be met for rapidly removing the smoke, dust, and gases produced by blasting and other mining operations.

During the early stages of mine development, fresh air was supplied to the working areas by a Joy, 8,000-c. f. m.-capacity, 1,000-series, Axivane blower, which was mounted at the collar of the main shaft and coursed fresh air down the shaft and to the working places through 20-inch-diameter sheet-metal ventilating pipe. Fresh air is now supplied by a Joy, 36-inch-diameter, 30,000-c. f. m.-capacity, 1,000-series, Axivane blower, installed directly over the collar of a ventilation shaft that is about 440 feet northeast of the main shaft (fig. 25). The blower is operated at 1,750 r. p. m. by a 30-horsepower, 440-volt, 3-phase motor, integral with the fan housing.

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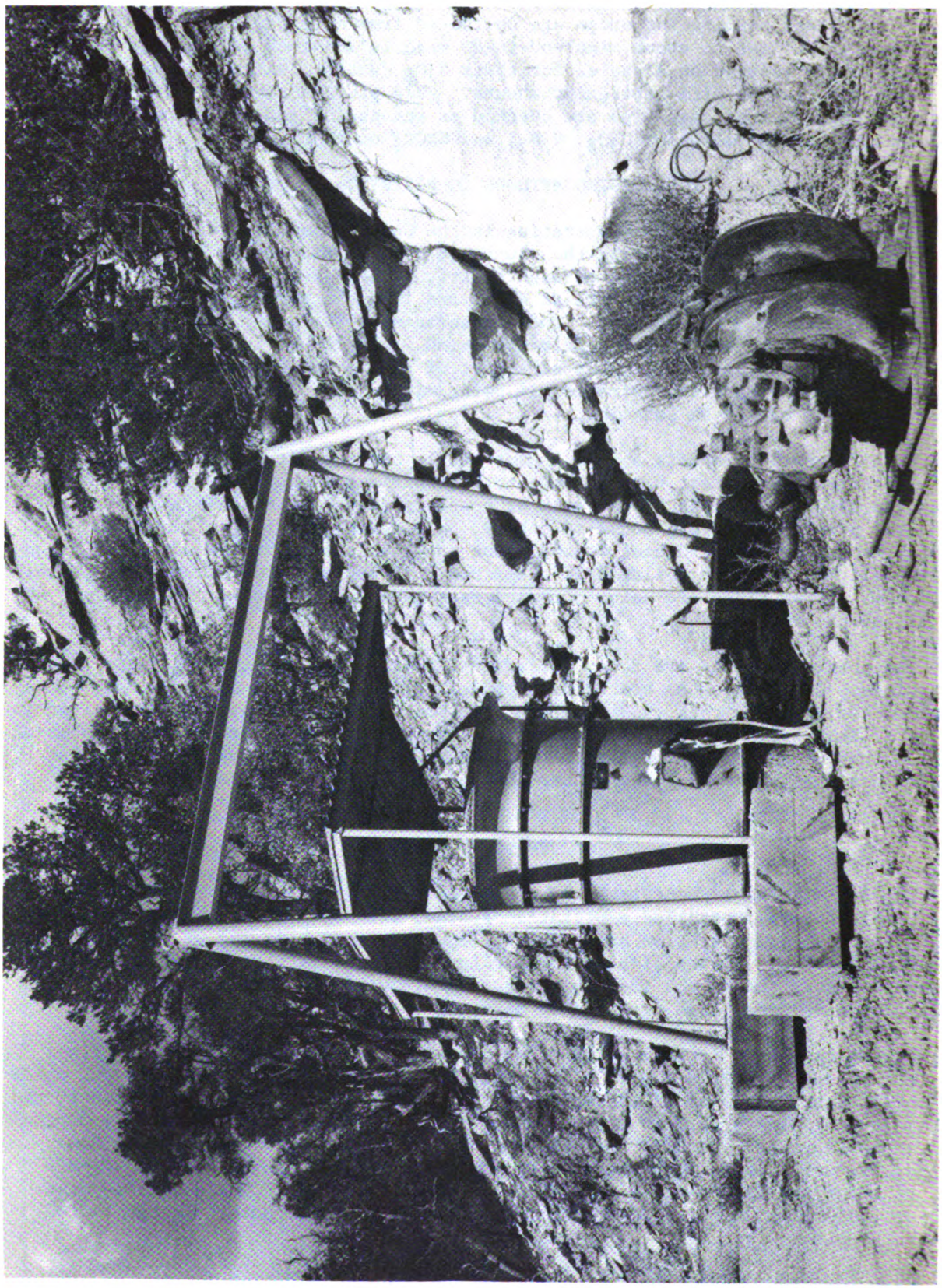

Figure 25. - 30,000-c. f. m. fan at collar of ventilation shaft.

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The ventilation shaft is circular, 36 inches in diameter, and 518 feet in depth. It extends to the main haulage level and is connected with the haulage drift by a short crosscut. The shaft was sunk by churn drilling (during the summer of 1955) after mine-air sampling in the expanding area of development indicated that airborne radioactive concentrations were exceeding the allowable limit. The shaft was drilled by contract drillers with a truck-mounted Bucyrus-Erie churn drill and paid for on a unit-cost basis. A 21-inch-diameter pilot hole was drilled then reamed to 36 inches with a 6-point reaming bit built around a 21-inch pilot stem. The contractor had trouble when the stem broke on the larger bit.

The collar of the shaft is concrete. The shaft is lined completely with cylindrical sections of sheet steel; the inside diameter of the steel casing is 34 inches; the bottom 150 feet is 7-gage steel, and the top 368 feet, 12-gage steel. The sheet-steel casing reduces the overall diameter by only 2 inches and allows maximum utilization of the cross-sectional area. The casing causes no shock loss, there is little friction to impede airflow, and the opening is fireproof. When the hole was bottomed, the steel casing was installed by welding cylindrical sections at the top and lowering the column as each new section was welded in place. As the casing was lowered, the outside was painted with asphaltic material to prevent rusting. Two transits, set a short distance from the shaft at an angle of 90° to each other, were used to insure that each new section was welded in line with the lower sections. The contract unit cost for drilling the churn-drill hole was $34 per foot and for lining the shaft $10.90 per foot.

Airflow is controlled by ventilation doors and auxiliary blowers. From the bottom of the ventilation shaft, fresh air passes through the crosscut and is diverted through the north and east haulage drifts and away from the main shaft by a ventilation door on the main-shaft side of the crosscut. Air currents travel up the manway compartments of the raises, are broadcast through the workings, and finally are exhausted through the main shaft. Ventilation doors in the dip inclines divert the air to the areas where it is needed. If air is not needed in an area serviced by a particular riase, a stopping is placed across the top of the raise. Air is diverted from the main aircourses to dead-end development headings by auxiliary blowers and ventilation pipe. Coppus Ventair TM-6 centrifugal blowers, powered by 5-horsepower motors, and 12- and 16-inch neoprene or metal ventilation pipe are used.

Samples of mine air are taken periodically at working places and checked radiometrically to determine the atmospheric concentration of the immediate daughters of radon gas. Mine air is sampled at least once a month; and the date, computed concentration of radon-daughter elements, ventilation conditions, and description of area sampled are recorded. This information is tabulated on a standard form for review by the State mine inspector, as required by an amendment to the General Safety Orders, effective January 1, 1956, issued by the Industrial Commission of Utah. This amendment states,

> The atmospheric concentration of the immediate daughters of radon should not exceed 300 MMCL (micro-microcuries per liter) as determined by a field method acceptable to the State Bureau of Mines (Industrial Commission), and every operator shall make a reasonable effort to approximate said standard.

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LABOR CLASSIFICATIONS, WAGES, AND CONTRACT BONUS SYSTEM

Labor Classifications and Wages

Two shifts are employed at the La Sal mine to work from 8 a.m. to 4 p.m. and from 6 p.m. to 2 a.m. The work schedule is 12 days on and 2 days off; the offdays fall on weekends. All employees, except the supervisory staff and equipment-maintenance crew, work alternate shifts every 2 weeks. The wage differential for working on the night shift is $0.06 per hour. When men are temporarily assigned to work on the graveyard shift (midnight to 8 a.m.), as might be necessary to complete important repair work, a shift differential of $0.08 per hour is paid.

Besides the supervisory staff, an average of 40 men is employed - 23 on the day shift and 17 on the night shift. The average number of men on each shift and the classifications and hourly base wages of the employees are shown in table 1.

TABLE 1. - Average number per shift, classifications, and hourly base wages of employees at La Sal mine

Labor classification	Base rate per hour	Number of men employed	
		Day shift	Night shift
Electrician[1/]	$2.45	1	
Mechanic, lead[1/]	2.45	1	
Mechanic	2.35	1	
Hoistman	2.25	1	1
Miner:			
Class A	2.25	10	10
Class B	2.15	2	2
Station tender	2.15	1	1
Motorman	2.15	2	2
Scraper operator	2.05	1	1
Utility man	2.05	1	
Laborer:			
Underground	1.85	1	
Surface	1.75	1	
Total	-	23	17

1/ Part-time employment at other Homestake Mining Co. operations in district.

Class A miners are contract miners concerned only with driving development headings and mining. An average of 10 contract miners is assigned to the crew on each shift. Class B miners are day's pay men who do various general tasks, such as install roof bolts and timber and construct scraper slides. The utility man on the day shift is the truckdriver. The station tender operates the slusher-pocket scraper, sharpens drill steel, tends the battery chargers, and handles equipment and supplies lowered or hoisted in the skips.

Six-day vacations are given to hourly wage employees for the first year of service and 1 additional day for each additional year of continuous service (to a maximum of 12 days). The company pays one-half the cost of membership in a national medical and surgical hospital-group plan for the miners and their families.

Contract Bonus System

To speed completion of the development program and to increase the tons produced from development headings per man-shift, a contract bonus system was initiated

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in March 1955. The incentive provided under the system has sped drifting, increased production, and decreased the cost per ton of ore. Payments to contractors are based on footage, the rates depending on the size of the excavation and the method and distance of haulage. The contract rates are based on those established earlier at the Little Beaver mine - another operation controlled by the Homestake Mining Co. A contract crew consists of men on both shifts, who work jointly under one contract to drive as many as six development headings.

The bonus plan for mining was begun in December 1955. A stope crew normally consists of 3 to 6 men divided between the 2 shifts. Contract bonus payments are based on the cubic feet of ore excavated each pay period.

Four to six contracts usually are in force at one time. Contract measurements are made semimonthly. The total bonus earned by a contract crew each semimonthly pay period is the difference between the total day's pay wages of the crew, calculated for each individual at his hourly base rate, and the total amount earned by the crew for work completed under its contract. The hourly bonus is the same for each man in a crew and equals the total bonus earned during the pay period, divided by the sum of hours worked by the men in the crew. The hourly bonus is then added to the hourly base wage of each man to determine his total hourly wage for the period. When a contract crew fails to earn an amount equal to its day's pay wages, it receives the latter. The contract-labor rates are listed below:

Development:	Contract rate per foot of advance
Scraper drifting: Includes installing pipe and ventilation lines but no track. Ore removed by scrapers.	
Drift size: 6 by 6 or 5 by 7 feet.	
Scraping distance, feet:	
0-100	$7.00
100-150	7.50
150-200	8.00
200-250	8.75
250-300	9.50
When scraper drifts are driven larger than basic size on instructions from the mine foreman, the above rates are increased as follows:	
Drift area, square feet:	
36-46	$0.50
47-57	1.00
58-68	1.50
69-79	2.00
When double scraping (using two units in relay) or scraping and then loading with a mechanical, track-mounted loader, the above rates are increased by $0.50.	
Sill drifting: Includes installation of track, pipe, and ventilation line.	

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	Contract rate per foot of advance
Drift size, feet:	
7 by 7	$8.75
7 by 8	9.75
Raising:	
Raise size, feet:	
5 by 5, open	8.00
7 by 8, open	9.75
6 by 13, 2-compartment, timbered	13.50

Mining:	Contract rate per cubic foot
Stoping: Includes drilling, blasting, and scraping to the nearest drawpoint.	
Scraping distance, feet:	
0-100	$0.097
100-150	.104
150-200	.111
200-250	.118
When double scraping or scraping and loading with track-mounted loader, the above rates are increased by $0.007.	
Slabbing: Includes mining, scraping, loading, and tramming. Widening drifts, etc., which cannot be classified as stoping	.104

The bonus earned by the contract crews depends partly on the increased effort required by the men who transport the ore from underground transfer points to the surface. To compensate the motormen, station tenders, and hoistmen for increasing mine production, the management awards to them an hourly bonus similiar to the bonuses earned by the contract crews. The total of the bonuses paid to the eight transport men is determined at one-third of the total of all bonuses of the contract crews. This amount, divided by the total hours worked by the eight men, is the hourly bonus added to each man's individual hourly base rate. When the value of work completed under a contract is less than the base-pay wages of the crew, the difference is subtracted from the amount on which the bonuses of the transport man is calculated, because it represents a decrease in tonnage transported.

SAFETY AND FIRST AID

Particular attention is given to the safety of employees. Every safety precaution is taken normal to orderly mine operation and as required by the General Safety Orders of the Industrial Commission of Utah. Danger signs are posted at the transformer station and at other installations where electrical hazards exist. The main underground powerlines are armored cable, and all power installations are grounded and equipped with circuit breakers. Danger signs are displayed at the explosives magazine.

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Seven is the maximum number of men permitted to ride in the skip at any one time. The man-trip hoisting or lowering speed is about half of normal speed. The hoist is equipped with a thrustor-type safety brake.

First-aid cabinets are maintained in the hoistroom, along with a basket-type stretcher and blankets.

The company keeps an ambulance at the La Sal shaft for emergencies.

SUMMARY OF COSTS

Name of mine: La Sal

Mining method: Panel retreating

Tons hoisted: 33,098 stope ore; 1,796 development ore; 2,941 development waste; 37,835 total

Period covered: October 1956 through February 1957

Summary of direct costs in units of labor, power, and supplies:

	Development[1]	Stoping[2]	Total[3]
Labor:			
Production, man-hour per ton:			
Drilling, blasting, barring	0.34	0.24	0.25
Scraping	.15	.17	.17
Loader-machine mucking	.05		.01
Hand mucking	.03	.01	.01
Roof bolting	.03	.03	.03
General	.13	.05	.06
Total	.73	.50	.53
Production ..... tons per man-shift	11.0	16.0	15.2
Other, underground, tons per man-shift:			
General, bull gang[4]			120.5
Haulage			73.5
Hoisting and caging			73.6
Supervision[5]			71.0
All labor, underground ...tons per man-shift			8.7
Surface labor, properly chargeable to underground operation .....tons per man-shift			62.1
All labor and supervision, surface and underground .....tons per man-shift			7.6

1/ Development unit costs were calculated on basis of 1,796 tons of development ore and 2,941 tons of development waste. Waste tons were included to show true costs, as the waste varies considerably each month.

2/ Stoping unit costs were calculated on 33,098 tons of stope ore.

3/ Total unit costs are prorated averages for total ore and waste produced from development and stoping.

4/ Miscellaneous underground work including long-hole drilling.

5/ Includes full time of foreman and night shifter and part time of engineer, sampler, asst. superintendent and superintendent chargeable to La Sal operation.

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Summary of direct costs in units of labor, power, and supplies: (Con.)

	Development[1/]	Stoping[2/]	Total[3/]
Labor: (Con.)			
Labor and supervision[6/] .... percent of total cost			61.75
Power and supplies:			
Explosives (40-percent dynamite) ... lb./ton	4.04	2.67	2.84
Power, total ..... kw.-hr./ton			13.29
Supplies and power[6/].. percent of total cost			38.25

See footnotes 1, 2, and 3 on p. 47.
6/ General office expense and taxes included.

Development costs:

Costs to sink the 572-foot shaft (rock section 8-1/2 by 18-1/2 feet) timbered in three 5- by 6-foot compartments.

Direct costs:

Item	Cost per foot
Labor	$81.83
Supervision	6.29
Timber	27.64
Explosives	5.96
Drill steel and bits	1.62
Air, water, and ventilation lines and fittings	5.69
Hanging rods and fittings	3.13
Shooting casing	1.11
Miscellaneous supplies and freight	9.06
Small tools	1.07
Equipment repair	1.61
Power generation and compressed air	6.56
Equipment rental	5.52
Total	157.09

Indirect costs:

Item	Cost per foot
Contract sinking fee	$23.01
Payroll taxes, workman's compensation, etc.	8.63
Miscellaneous surface expense	1.40
Total	33.04

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